DRILLING AND BLASTING OF ROCKS

CARLOS LOPEZ JIMENO
Project Director for EPM, S.A.

EMILIO LOPEZ JIMENO

FRANCISCO JAVIER AYALA CARCEDO
Project Director for ITGE

Translated by
YVONNE VISSER DE RAMIRO

A.A. BALKEMA / ROTTERDAM / BROOKFIELD / 1995

This work has been totally financed by the Geomining Technological Institute of Spain under contract with the E.P.M., S.A. Company (Estudios y Proyectos Mineros, S.A.).

Authorization to photocopy items for internal or personal use, or the internal or personal use of specific clients, is granted by A.A.Balkema, Rotterdam, provided that the base fee of US$1.50 per copy, plus US$0.10 per page is paid directly to Copyright Clearance Center, 222 Rosewood Drive, Danvers, MA 01923, USA. For those organizations that have been granted a photocopy license by CCC, a separate system of payment has been arranged. The fee code for users of the Transactional Reporting Service is: 90 5410 199 7/95 US$1.50 + US$0.10.

Original text:
Manual de perforacion y voladura de rocas
© 1987 Instituto Geologico y Minero de España

Revised and updated edition in English:
© 1995 A.A.Balkema, P.O.Box 1675, 3000 BR Rotterdam, Netherlands (Fax: +31.10.4135947)

ISBN 90 5410 199 7

Distributed in USA & Canada by:
A.A.Balkema Publishers, Old Post Road, Brookfield, VT 05036, USA (Fax: 802.276.3837)

Printed in the Netherlands

Contents

FOREWORD IX

PREFACE XI

ACKNOWLEDGEMENTS XIII

1 ROCK DRILLING METHODS 1
- 1.1 Introduction 1
- 1.2 Types of drilling operations used in rock breakage 1
- 1.3 Fields of application for the different drilling methods 1
- 1.4 Classification of the rocks and their principal physical properties 3
- References 7

2 ROTARY PERCUSSIVE DRILLING 8
- 2.1 Introduction 8
- 2.2 Fundamentals of rotary percussive drilling 8
- 2.3 Top hammer drilling 10
- 2.4 Drilling with down the hole hammer 16
- 2.5 Advance systems 18
- 2.6 Mounting systems 19
- 2.7 Dust collectors 28
- 2.8 Inclination instruments 30
- 2.9 Penetration rate 30
- 2.10 Average penetration rate 33
- 2.11 Calculation of drilling costs 34
- References 35

3 ROTARY PERCUSSIVE DRILLING ACCESSORIES 36
- 3.1 Introduction 36
- 3.2 Types of threads 36
- 3.3 Shank adaptors 37
- 3.4 Drill steel 38
- 3.5 Couplings 40
- 3.6 Drill bits 41
- 3.7 Calculation of the necessary drilling accessories 43
- 3.8 Care and maintenance of the bits 43
- 3.9 Care and maintenance of drill steel 47
- 3.10 Guide for identifying accessory failure and its causes 47
- References 47

4 ROTARY DRILLING WITH ROLLING TRICONE BITS 48
- 4.1 Introduction 48
- 4.2 Mounting and propulsion systems 48
- 4.3 Power sources 48
- 4.4 Rotation systems 49
- 4.5 Pulldown/hoisting systems 50
- 4.6 Mast and pipe changer 51
- 4.7 Control cabin 52
- 4.8 System for flushing drill cuttings 52
- 4.9 Drill string 53
- 4.10 Auxiliary elements 55
- 4.11 Operative practice. Drilling parameters 56
- 4.12 Penetration rate 59
- 4.13 Calculation of drilling costs 61
- References 63

5 ROLLING CONE ROCK BITS 64
- 5.1 Rolling cone rock bits 64
- 5.2 Major components and design features 64
- 5.3 The metallurgy of rolling cone rock bits 66
- 5.4 Types of rolling cone bits 67
- 5.5 Bit type selection 67
- 5.6 Effects of the operating parameters on the rolling cone bits 69
- 5.7 Nozzle selection 69
- 5.8 Evaluation of dull rolling cones 70
- 5.9 Example of roller tricone bit selection 71
- 5.10 IDAC Codes 71
- References 71

6 ROTARY DRILLING WITH CUTTING ACTION 72
- 6.1 Introduction 72
- 6.2 Fundamentals of drilling with cutting action 72
- 6.3 Flushing of drill cuttings 75
- 6.4 Cutting tools 75
- References 76

7 SPECIAL DRILLING METHODS AND MOUNTING SYSTEMS 77
- 7.1 Introduction 77
- 7.2 Drilling through overburden 77
- 7.3 Shaft sinking 79
- 7.4 Raise driving 80

7.5 Jet piercing 83
7.6 Water-jet drilling 84
7.7 Drilling ornamental rock 85
References 86

8 COMPRESSORS 87
8.1 Introduction 87
8.2 Types of compressors 87
8.3 Drive 89
8.4 Auxiliary elements 89
8.5 Calculating pressure drops 90
References 91

9 THERMOCHEMISTRY OF EXPLOSIVES AND THE DETONATION PROCESS 92
9.1 Introduction 92
9.2 Deflagration and detonation 92
9.3 Detonation process of an explosive 93
9.4 Thermochemistry of the explosives 94
9.5 Heat of explosion 94
9.6 Oxygen balance 95
9.7 Volume of explosion 95
9.8 Minimum energy available 96
9.9 Temperature of the explosion 96
9.10 Pressure of the explosion 96
References 97

10 PROPERTIES OF EXPLOSIVES 98
10.1 Introduction 98
10.2 Strength and energy 98
10.3 Detonation velocity 101
10.4 Density 102
10.5 Detonation pressure 102
10.6 Stability 102
10.7 Water resistance 102
10.8 Sensitivity 102
10.9 Detonation transmission 103
10.10 Desensitization 104
10.11 Resistance to low temperatures 104
10.12 Fumes 104
References 105

11 INDUSTRIAL EXPLOSIVES 106
11.1 Introduction 106
11.2 Dry blasting agents 106
11.3 Slurries 110
11.4 Emulsions 111
11.5 Heavy ANFO 113
11.6 Gelatin dynamites 115
11.7 Granular dynamite 115
11.8 Permissible explosives 116
11.9 Blackpowders 116
11.10 Two-component explosives 117
11.11 Explosives commercialized in Spain 117
References 117

12 EXPLOSIVE SELECTION CRITERIA 119
12.1 Introduction 119
12.2 Explosive cost 119
12.3 Charge diameter 119
12.4 Rock characteristics 119
12.5 Volume of rock to be blasted 121
12.6 Atmospheric conditions 121
12.7 Presence of water 121
12.8 Environmental problems 121
12.9 Fumes 122
12.10 Safety conditions 122
12.11 Explosive atmospheres 122
12.12 Supply problems 122
References 122

13 BLASTING ACCESSORIES 123
13.1 Introduction 123
13.2 Nonelectric initiation systems 123
13.3 Electric initiation systems 127
13.4 Sources of energy 130
13.5 Other accessories 132
References 135

14 INITIATION AND PRIMING SYSTEMS 136
14.1 Introduction 136
14.2 Priming and boostering bulk ANFO-type blasting agents 136
14.3 Priming cartridge ANFO type blasting agents 138
14.4 Priming pumped or poured slurry and emulsion blasting agents 139
14.5 Priming cartridged watergel and emulsion blasting agents 140
14.6 Location of primers 140
14.7 Priming conventional cartridged explosives 143
References 143

15 MECHANIZED SYSTEMS FOR CHARGING AND DEWATERING BLASTHOLES 144
15.1 Introduction 144
15.2 Mechanized blasthole charging systems 144
15.3 Blasthole dewatering systems 152
References 153

16 MECHANISMS OF ROCK BREAKAGE 154
16.1 Introduction 154
16.2 Rock breakage mechanisms 154
16.3 Transmission of the strain wave through the rock mass 156
16.4 Energetic yield of the blastings 157
References 159

17 ROCK AND ROCK MASS PROPERTIES AND THEIR INFLUENCE ON THE RESULTS OF BLASTING 160
17.1 Introduction 160
17.2 Rock properties 160
17.3 Properties of the rock mass 161
References 166

18 CHARACTERIZATION OF THE ROCK MASSES FOR BLAST DESIGNING 167
- 18.1 Introduction 167
- 18.2 Diamond drilling with core recovery and geomechanic testing 167
- 18.3 Characteristics of the joint systems 167
- 18.4 Seismic survey 170
- 18.5 Geophysical techniques to obtain rock mass data 170
- 18.6 Logging of production blastholes 170
- 18.7 Characterization of the rock mass during blasthole drilling 171
- 18.8 The attempt to correlate drilling indexes with the blasting design parameters 174
- 18.9 System of drilling data management in actual time 177
- References 178

19 CONTROLLABLE PARAMETERS OF BLASTING 179
- 19.1 Introduction 179
- 19.2 Blasthole diameter 179
- 19.3 Height of bench 181
- 19.4 Blasthole inclination 181
- 19.5 Stemming length 182
- 19.6 Subdrilling 182
- 19.7 Burden and spacing 183
- 19.8 Blasthole patterns 183
- 19.9 Geometry of the free face 184
- 19.10 Size and shape of the blast 185
- 19.11 Available expansion volume 186
- 19.12 Charge configuration 186
- 19.13 Decoupling of the charges 186
- 19.14 Explosives 187
- 19.15 Distribution of explosives in the blastholes 187
- 19.16 Powder factor 188
- 19.17 Initiation and priming 188
- 19.18 Delay timing and initiation sequences 188
- 19.19 Influence of loading equipment on the design of the blasts 189
- 19.20 Specific drilling 189
- 19.21 Blasthole deviation 190
- References 190

20 BENCH BLASTING 191
- 20.1 Introduction 191
- 20.2 Small diameter bench blasting 191
- 20.3 Large diameter blasting 193
- 20.4 Bench blasting with horizontal blastholes 195
- 20.5 Rip-rap production blasting 195
- 20.6 Cast blasting 196
- Appendix 1: Formulas to calculate bench blasting patterns 199
- References 203

21 BLASTING IN OTHER SURFACE OPERATIONS 205
- 21.1 Introduction 205
- 21.2 Excavations for highways and railways 205
- 21.3 Trench blasting 208
- 21.4 Ramp blasting (sinking cut) 210
- 21.5 Blasting for ground leveling 212
- 21.6 Blastings for foundations 213
- 21.7 Mini-hole blasting 214
- 21.8 Preblastings 215
- References 216

22 BLASTING FOR TUNNELS AND DRIFTS 217
- 22.1 Introduction 217
- 22.2 Advance systems 217
- 22.3 Blasting patterns for tunnels 218
- 22.4 Types of cuts and calculation of the blasts 219
- 22.5 Equipment for marking out drilling patterns 230
- References 230

23 SHAFT SINKING AND RAISE DRIVING 231
- 23.1 Introduction 231
- 23.2 Shaft sinking 231
- 23.3 Raise driving 232
- References 237

24 UNDERGROUND PRODUCTION BLASTING IN MINING AND CIVIL ENGINEERING 239
- 24.1 Introduction 239
- 24.2 Crater retreat method 239
- 24.3 Longhole method 243
- 24.4 Sublevel stoping with blastholes in fan pattern 245
- 24.5 Room and pillar mining 248
- 24.6 Cut and fill mining 248
- 24.7 Underground chambers in civil engineering projects 249
- References 251

25 CONTOUR BLASTING 252
- 25.1 Introduction 252
- 25.2 Mechanisms responsable for overbreak 252
- 25.3 The theory of contour blasting 253
- 25.4 Types of contour blasts 254
- 25.5 The parameters that intervene in a contour blasting 256
- 25.6 Tendencies in the field of contour blasting 264
- 25.7 Evaluation of the results 267
- 25.8 Example 268
- 25.9 Extraction of ornamental rock with contour blasting 268
- References 270

26 UNDERWATER BLASTING 272
- 26.1 Introduction 272
- 26.2 Methods of execution 272
- 26.3 Calculations for charges and drilling patterns 247
- 26.4 Charging the blastholes and priming systems 275
- 26.5 Types of explosives 276
- 26.6 Environmental effects associated with underwater blastings 276

18 CHARACTERIZATION OF THE ROCK MASSES FOR BLAST DESIGNING 167
18.1 Introduction 167
18.2 Diamond drilling with core recovery and geomechanic testing 167
18.3 Characteristics of the joint systems 167
18.4 Seismic survey 170
18.5 Geophysical techniques to obtain rock mass data 170
18.6 Logging of production blastholes 170
18.7 Characterization of the rock mass during blasthole drilling 171
18.8 The attempt to correlate drilling indexes with the blasting design parameters 174
18.9 System of drilling data management in actual time 177
References 178

19 CONTROLLABLE PARAMETERS OF BLASTING 179
19.1 Introduction 179
19.2 Blasthole diameter 179
19.3 Height of bench 181
19.4 Blasthole inclination 181
19.5 Stemming length 182
19.6 Subdrilling 182
19.7 Burden and spacing 183
19.8 Blasthole patterns 183
19.9 Geometry of the free face 184
19.10 Size and shape of the blast 185
19.11 Available expansion volume 186
19.12 Charge configuration 186
19.13 Decoupling of the charges 186
19.14 Explosives 187
19.15 Distribution of explosives in the blastholes 187
19.16 Powder factor 188
19.17 Initiation and priming 188
19.18 Delay timing and initiation sequences 188
19.19 Influence of loading equipment on the design of the blasts 189
19.20 Specific drilling 189
19.21 Blasthole deviation 190
References 190

20 BENCH BLASTING 191
20.1 Introduction 191
20.2 Small diameter bench blasting 191
20.3 Large diameter blasting 193
20.4 Bench blasting with horizontal blastholes 195
20.5 Rip-rap production blasting 195
20.6 Cast blasting 196
Appendix 1: Formulas to calculate bench blasting patterns 199
References 203

21 BLASTING IN OTHER SURFACE OPERATIONS 205
21.1 Introduction 205
21.2 Excavations for highways and railways 205
21.3 Trench blasting 208
21.4 Ramp blasting (sinking cut) 210
21.5 Blasting for ground leveling 212
21.6 Blastings for foundations 213
21.7 Mini-hole blasting 214
21.8 Preblastings 215
References 216

22 BLASTING FOR TUNNELS AND DRIFTS 217
22.1 Introduction 217
22.2 Advance systems 217
22.3 Blasting patterns for tunnels 218
22.4 Types of cuts and calculation of the blasts 219
22.5 Equipment for marking out drilling patterns 230
References 230

23 SHAFT SINKING AND RAISE DRIVING 231
23.1 Introduction 231
23.2 Shaft sinking 231
23.3 Raise driving 232
References 237

24 UNDERGROUND PRODUCTION BLASTING IN MINING AND CIVIL ENGINEERING 239
24.1 Introduction 239
24.2 Crater retreat method 239
24.3 Longhole method 243
24.4 Sublevel stoping with blastholes in fan pattern 245
24.5 Room and pillar mining 248
24.6 Cut and fill mining 248
24.7 Underground chambers in civil engineering projects 249
References 251

25 CONTOUR BLASTING 252
25.1 Introduction 252
25.2 Mechanisms responsable for overbreak 252
25.3 The theory of contour blasting 253
25.4 Types of contour blasts 254
25.5 The parameters that intervene in a contour blasting 256
25.6 Tendencies in the field of contour blasting 264
25.7 Evaluation of the results 267
25.8 Example 268
25.9 Extraction of ornamental rock with contour blasting 268
References 270

26 UNDERWATER BLASTING 272
26.1 Introduction 272
26.2 Methods of execution 272
26.3 Calculations for charges and drilling patterns 247
26.4 Charging the blastholes and priming systems 275
26.5 Types of explosives 276
26.6 Environmental effects associated with underwater blastings 276

26.7 Shaped or concussion charges 278
References 279

27 INITIATION SEQUENCE AND DELAY TIMING 281
27.1 Introduction 281
27.2 Single-row delayed blast 281
27.3 Multi-row sequenced bench blastings 282
27.4 Bench blasting sequences for underground stopes 284
27.5 Delay timings 285
27.6 Underground blasts in tunnels and drifts 287
References 289

28 EVALUATION OF BLAST RESULTS 290
28.1 Introduction 290
28.2 Fragmentation and swelling of the muckpile 290
28.3 Geometry of the muckpile, its height and displacement 293
28.4 Condition of the remaining mass 294
28.5 Analysis of the bench floor 295
28.6 Boulders in the muckpile 295
28.7 Vibrations and airblast 296
28.8 Profiles of underground excavations 296
28.9 Conclusions 296
References 296

29 SECONDARY FRAGMENTATION AND SPECIAL BLASTINGS 299
29.1 Introduction 299
29.2 Pop shooting 299
29.3 Secondary breakage by mechanical means and special methods 299
29.4 Special blastings 302
References 304

30 PLANNING THE WORK OF DRILLING AND BLASTING 305
30.1 Introduction 305
30.2 Factors that have influence on the planning of drillling and blasting 305
30.3 Planning the stages of excavation 308
References 311

31 STRUCTURE AND BUILDING DEMOLITION 312
31.1 Introduction 312
31.2 Drilling diameters and types of explosives 312
31.3 Demolition of structural elements 313
31.4 Demolition of structures 316
31.5 Demolition of buildings 320
31.6 Demolition of steel structures 322
References 322

32 OPTIMIZING COSTS OF FRAGMENTATION WITH DRILLING AND BLASTING 323
32.1 Introduction 323
32.2 Economical aspects of drilling and blasting 323
32.3 Model for determining cost optimization 325
32.4 Predicting the fragmentation 326
32.5 Probabilistic analysis optimization model 331
References 331

33 LAND VIBRATIONS, AIR BLAST AND THEIR CONTROL 333
33.1 Introduction 333
33.2 Parameters which affect vibration characteristics 333
33.3 Characteristics of ground vibrations 337
33.4 Air blast characteristics 339
33.5 Instrumentation for recording and analyzing vibrations and air blast 340
33.6 Calculators of propogation laws for land and air vibrations 342
33.7 Studies of vibration and air blast 346
33.8 Damage prevention criteria for buildings 350
33.9 Effects of vibrations and air blast on people 357
33.10 Effects of vibrations on rock masses 358
33.11 Effect of vibrations on freshly poured concrete 360
33.12 Recommendations for reducing ground vibration and air blast levels 361
References 364

34 FLYROCKS AND THEIR CONTROL 366
34.1 Introduction 366
34.2 Models to calculate the throw of flyrock 366
34.3 Coverings 368
34.4 Recommendations for carrying out bench blastings 370
References 370

35 SAFETY MEASURES FOR DRILLING AND BLASTING OPERATIONS 371
35.1 Introduction 371
35.2 Blasthole drilling 371
35.3 Blastings 375
References 381

CONVERSION FACTORS 382

GENERAL INFORMATION, WEIGHT OF MATERIALS 383

GLOSSARY 385

SUBJECT INDEX 389

Foreword

During the past two decades, there have been numerous technical contributions which have brought a better understanding of rock fragmentation with explosives, an improvement in drilling equipment and a noticeable evolution in the development of new explosives and blasting accessories. The Geomining Technological Institute of Spain (ITGE), aware of this progress and of the importance which the breakage process has acquired in mining and civil engineering projects, has considered the publication of a 'Rock Drilling and Blasting Handbook' of great interest.

This handbook was conceived with integration in mind, as the systems and machines of drilling, the types and characteristics of explosives and the methods for calculating the blasts are treated together, without ever forgetting that these breakage operations form part of a macrosystem and that the results obtained by them influence the production and economy of the whole exploitation or construction process. At the same time, the objectives and contents of this handbook contribute to improved safety in mining.

There are very few similar works in other languages, and certainly none other in Spanish.

We sincerely hope that this handbook, which brings together practical and theoretical aspects, will be of use to all engineers who work with drilling and blasting as a rock breakage method.

Camilo Caride de Liñan
Director of the Geomining Technological Institute of Spain

Preface

Rock breakage with explosives has existed since the XVII century when black powder came into use in mining, rapidly becoming one of the most popular methods. The important historical events which have marked an era were the invention of dynamite by Alfred Nobel in 1867, the use of ANFO starting in 1955, the development of slurries from the late fifties on and, lastly, the preparation of blasting agents such as emulsions, heavy ANFO, etc., which are still in evolution.

At the same time, blasthole drilling progressed with such decisive events as the the use of compressed air as the source of energy in rotary percussive rigs in 1861, the use of large rotary drills and of down-the-hole hammers in the fifties and the development of hydraulic hammers in the late seventies.

However, rock blasting was always considered, until recently, as an *art* born from the skill and experience of the blasters. Now it has become a technique based on scientific principles derived from knowledge of the action of explosives, the mechanisms of breakage and the geomechanic properties of the rock masses.

The purpose of this handbook is to give basic knowledge of the drilling systems, the types of available explosives and accessories and the parameters that intervene in blast designing, whether controllable or not.

The handbook is primarily meant for students of the Technical Schools, to be used as a textbook, and for all professionals who are involved with explosives in mining operations and civil engineering projects.

Carlos and Emilio Lopez Jimeno

This handbook was written by the following engineers:

Carlos Lopez Jimeno, Doctor of Mining Engineering, Project Director for EPM., S.A.
Emilio Lopez Jimeno, Doctor of Mining Engineering.
Francisco Javier Ayala Carcedo, Doctor of Mining Engineering, Project Director for ITGE.

Translated by: Yvonne Visser de Ramiro

This work has been totally financed by the Geomining Technological Institute of Spain under contract with the EPM, S.A. Company (Estudios y Proyectos Mineros, S.A.).

Acknowledgements

The authors wish to express their most sincere gratitude to the following experts, companies and official organisms for their collaboration and release of technical material as well as permission to reproduce certain data and figures.

Amerind-Mackissic, Inc.: G.J. Knotts
Amos L. Dolby Co.: J. Petrunyak
Applex:S.O. Olofsson
Atlas Copco S.A.E.: F. Menéndez
Atlas Powder Company: V.A. Sterner, L. Osen & P.M. Miller
Atlas Powder International: J. García Milla
Bauer, Calder & Workman, Inc.: J.L. Workman & A. Bauer (†)
Bill Lane Inc.: W.C. Lane
Blasting & Mining Consultants, Inc.: J. Ludwiczak
Bucyrys Erie Co.: J.D. Nelmark & G. Rekoske
Bendesanstalt für Geowissenschaften und Rohstdffe: R. Lüdeling
Canmet: G. Larocque
Cil Inc.: S. Chung, B. Mohanty, K.C. Joyce, P.R. Day, W.K. Webster, D. Dayphinais, I. Huss & K.R. Sharpe
Cominco Ltd.: W. Russell
Crowsnest Resources Ltd.: R.A. Reipas
David, S. Robertson & Associates Inc.: C. Davenport
Dupont Canada: D. Tansey
E.I. Du Pont De Nemours & Co.: P.D. Porter, B.L. Glenn, J.R. Knudson & A.B. Andrews
Entrecanales y Tavora, S.A.: J. Aznar
Gardner Denver Mining and Construction Group
Geovanca: R. Ucar
Golder Associates: T.N. Hagan, E. Hoek & Guy Le Bell
Hullera Vasco Leonesa: E. Castells
Hydro-Quebec: P. Lacomte
Iberduero, S.A.: J. Fora
ICI Australia Operations Pty Ltd.: G. Harries, J.K. Mercer & G.G. Paine
Ilmeg: S. Johansson
Ingersoll Rand
Instituto Tecnológico Geominero de España: F.J. Ayala & M. Abad
Instituto Superior Técnico de Lisboa: C. Dinis Da Gama
Ireco Canada Inc.: L. de Couteur
Irish Industrial Explosives, Ltd.: J.P. Higgins
Julius Kruttschnitt Mineral Research Centre, University of Queensland: C.K. Mckenzie & K.E. Mathews
Kaiser Engineers, Inc.: G.V. Borquez
Kemira Oy
Kenneth Medearis Associates: K. Medearis
Kometa Oy: R. Ikola
Kontinitro A.G.
L.C. Lang & Associates, Inc.: L.C.Lang
Lewis L. Oriard, Inc.: L.L.Oriard
LKAB: L. Hermansson
Martin Marietta Laboratories: D.A. Anderson & S.R. Winzer
McGill University: R.F. Favreau, R.R. MacLachlan, W. Comeau & J.C. Leighton
Michigan Technological University: F.O. Otuonye
New Jersey Institute of Technology: W. Konon
Nitro Consult, A.B.: I. Hansson
Nitro Nobel AB: B. Larsson, P.A. Persson, M. Landberg & G. Lande
Nobel's Explosives Company Limited: M.J. Ball
The Norwegian Institute of Technology: K. Nielsen
The Ohio State University: R.G. Lundquist
Oy Forcit
Palabora Mining Co.: G.P. Fauquier
Petromin: V. Cobeña
Precision Blasting Services: C.J. Konya
Queen's University: P.N. Calder
Reed Mining Tools, Inc.: M. Suárez
Richard L. Ash & Associates: R.L. Ash
Rietspruit Mining Co.: K.I. Macdonald
Societa Esplosivi Industriali S.P.A.: G. Calarco & G. Berta
Strømme: A.M. Heltzen
Thermex Energy Corporation: R.C. Paddock
T. Peal, S.A.: J. Alonso & R.Arnaiz
Union Española de Explosivos: R. Blanco
University of Missouri Rolla: P.N. Worsey, R.R. Rollins & N.S. Smith
U.S. Bureau of Mines Twin Cities. Research Center: L.R. Fletcher

At the same time we would also like to acknowledge the drawings and photography done by Jose María de Salas and the corrections made by Carlos Ramiro Visser.

CHAPTER 1

Rock drilling methods

1.1 INTRODUCTION

Rock drilling, in the field of blasting, is the first operation carried out and its purpose is to open holes, with the adequate geometry and distribution within the rock masses, where the explosive charges will be placed along with their initiating devices.

The systems of rock drilling that have been developed and classified according to their order of present day applicability are:

- *Mechanical:* Percussion, rotary, rotary-percussion.
- *Thermal:* Flame, plasma, hot fluid, Freezing.
- *Hydraulic:* Jet, erosion, cavitation.
- *Sonic:* High frequency vibration.
- *Chemical:* microblast, dissolution.
- *Electrical:* Electric arc, magnetic induction.
- *Seismic:* Laser ray.
- *Nuclear:* Fusion, fission.

Even though there is an enormous variety of possible rock drilling systems, in mining and civil engineering drilling is presently carried out, almost exclusively, by mechanical energy. Therefore, in this handbook only the mechanical means will be discussed, reviewing the fundalmentals, tools and equipment for each of them.

The main components of a drilling system of this type are: the drilling rig which is the source of mechanical energy, the drill steel which is the means of transmitting that energy, the bit which is the tool that exercises that energy upon the rock, and the flushing air that cleans out and evacuates the drilling cuttings and waste produced.

1.2 TYPES OF DRILLING OPERATIONS USED IN ROCK BREAKAGE

Within the large variety of excavations using explosives, numerous machines have been developed which can be classified in two types of drilling procedures:

– *Manual drilling.* This is carried out with light equipment that is hand held by the drillers. It is used in small operations where, due to the size, other machinery cannot be used or its cost is not justified.

– *Mechanized drilling.* The drilling equipment is mounted upon rigs with which the operator can control all drilling parameters from a comfortable position. These structures or chasis can themselves be mounted on wheels or tracks and either be self-propelled or towable.

On the other hand, the types of work, in surface as well as in underground operations, can be classified in the following groups:

– *Bench drilling.* This is the best method for rock blasting as a free face is available for the projection of material and it allows work to be systemized.

It is used in surface projects as well as in underground operations, usually with vertical blastholes, although horizontal holes can be drilled on occasion.

– *Drilling for drifting and tunnelling.* An initial cavity or cut must be opened towards which the rest of the fragmented rock from the other charges is directed. Blasthole drilling can be carried out with hand held drills, but the trend is towards total mechanization, using jumbos with one or various booms.

– *Production drilling.* This term is used in mining operations, fundamentally underground, to describe the labors of ore extraction. The equipment and methods used vary with the exploitation systems, having the common factor of little available space in the drifts for blasthole drilling.

– *Drilling for raises.* In many underground and civil engineering projects it is necessary to open raises. Although there is a tendency to apply the Raise Boring method, still today the long blasthole method is used as well as other special drilling systems combined with blasting.

– *Drilling rocks with overburden.* The drilling of rock masses which are covered with beds of unconsolidated materials calls for special drilling methods with casing. This method is also used in underwater operations.

– *Rock supports.* In many underground operations and sometimes in surface ones it is necessary to support the rocks by means of bolting or cementing cables, in which drilling is the first phase.

1.3 FIELDS OF APPLICATION FOR THE DIFFERENT DRILLING METHODS

The two most used mechanical drilling methods are rotary-percussion and rotary.

– *Rotary-percussive methods.* These are the most frequently used in all types of rocks, the top hammer as well as the down-the-hole hammer.

– *Rotary methods.* These are subdivided into two groups, depending upon if the penetration is carried out by crushing, with tricones or by cut with drag bits. The first system is used in medium to hard rocks, and the second in soft rocks.

By taking into account the compressive strength of the rocks and the drilling diameter, the fields of application of the different methods can be defined as reflected in Fig. 1.1.

On the other hand, depending upon the type of mining or civil engineering surface project, the most common equipment and diameters for bench blastings are indicated in Fig. 1.2.

In the same manner, the most frequently used equipment for the different underground mining methods and the characteristic drilling data are indicated in Fig. 1.3.

Other criteria to be accounted for in the selection of drilling equipment are: cost, mechanical design, maintenance and service, operative capacity, adaptability to equipment of the exploitation, and the work area conditions (accessability, type of rock, sources of energy, etc.).

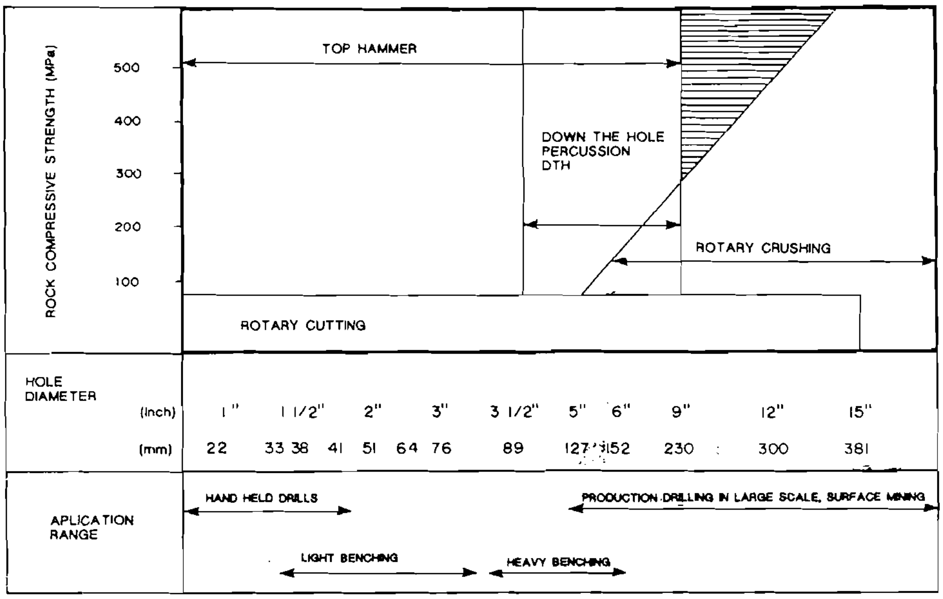


Fig. 1.1 Fields of application for drilling methods as function of the compressive strength of the rocks and the diameters of the blastholes.

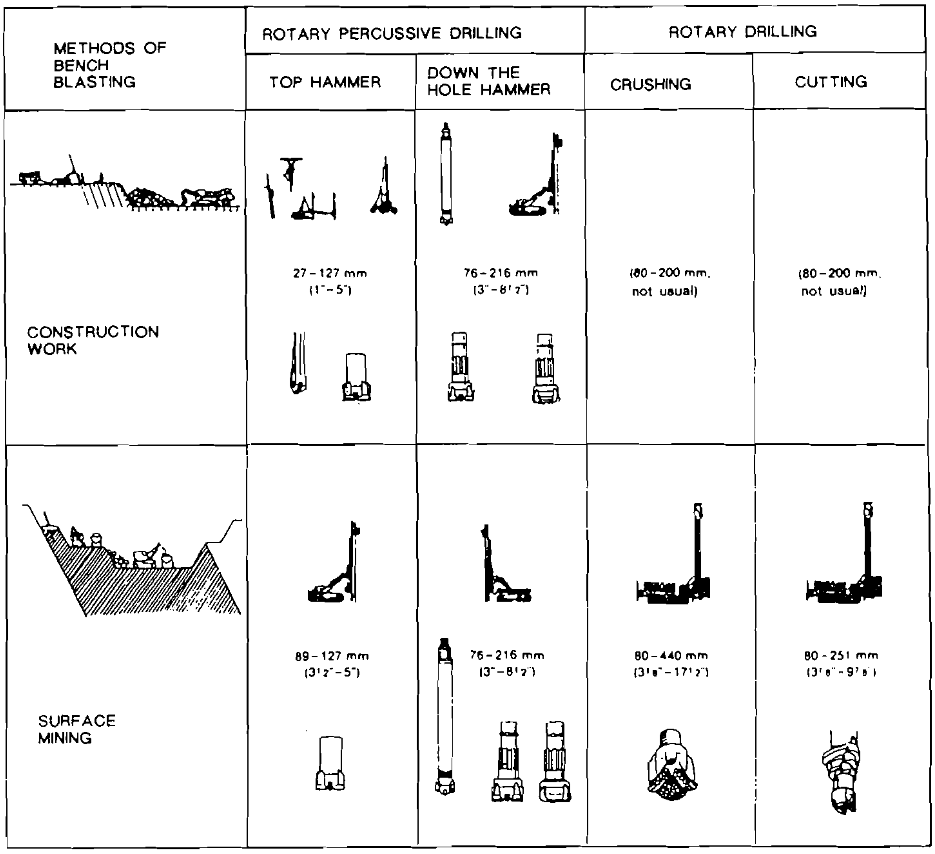
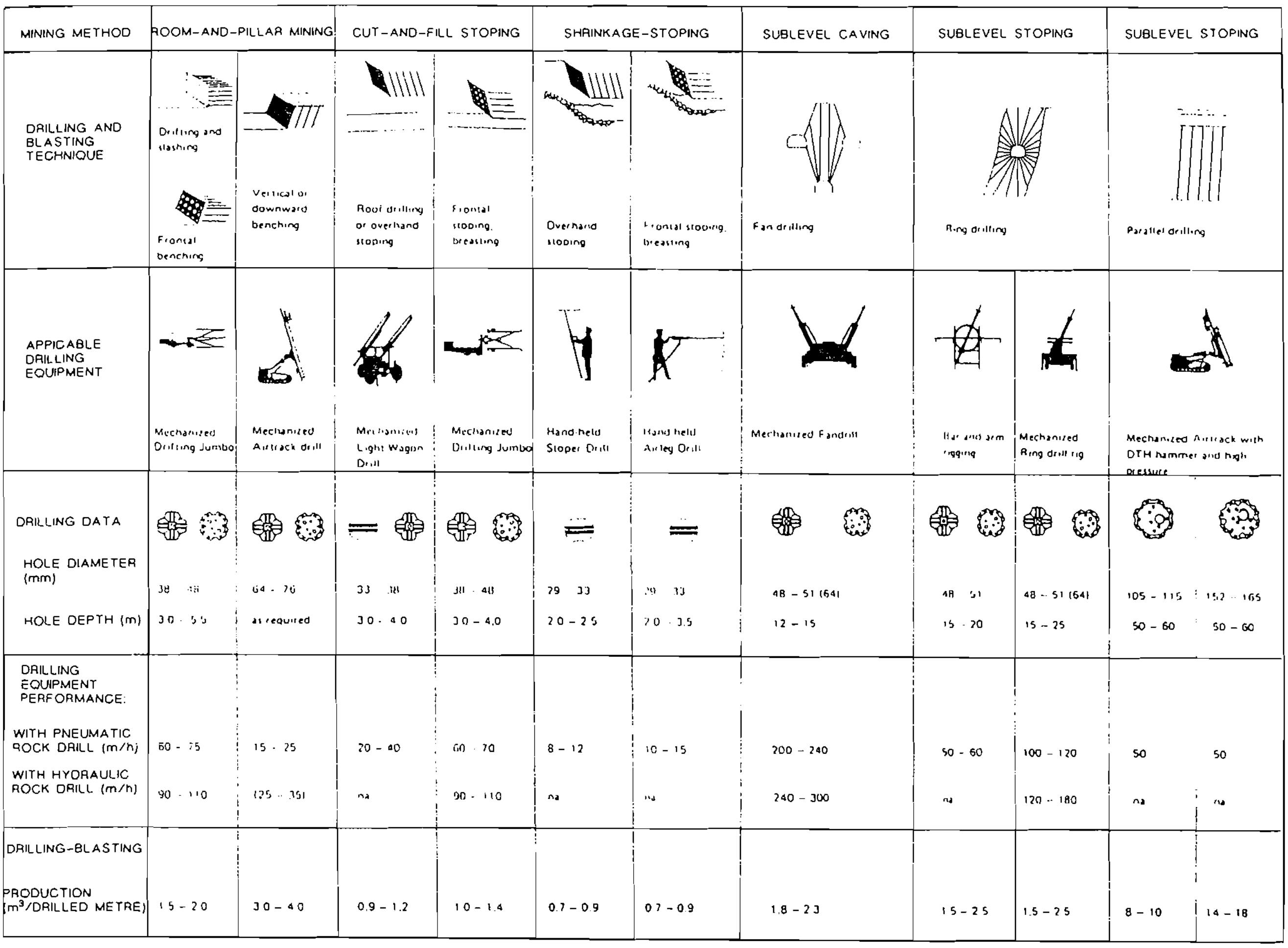


Fig. 1.2 Drilling methods for surface operations (Atlas Copco).



MINING METHOD	ROOM-AND-PILLAR MINING		CUT-AND-FILL STOPING		SHRINKAGE-STOPING		SUBLEVEL CAVING	SUBLEVEL STOPING		SUBLEVEL STOPING	
DRILLING AND BLASTING TECHNIQUE	Drifting and slashing Frontal benching	Vertical or downward benching	Roof drilling or overhand stoping	Frontal stoping, breasting	Overhand stoping	Frontal stoping, breasting	Fan drilling	Ring drilling		Parallel drilling	
APPICABLE DRILLING EQUIPMENT	Mechanized Drifting Jumbo	Mechanized Airtrack drill	Mechanized Light Wagon Drill	Mechanized Drilling Jumbo	Hand-held Stoper Drill	Hand held Airleg Drill	Mechanized Fandrill	Bar and arm rigging	Mechanized Ring drill rig	Mechanized Airtrack with DTH hammer and high pressure	
DRILLING DATA											
HOLE DIAMETER (mm)	38 – 48	64 – 76	33 – 38	38 – 48	29 – 33	29 – 33	48 – 51 (64)	48 – 51	48 – 51 (64)	105 – 115	152 – 165
HOLE DEPTH (m)	3.0 – 5.5	as required	3.0 – 4.0	3.0 – 4.0	2.0 – 2.5	2.0 – 3.5	12 – 15	15 – 20	15 – 25	50 – 60	50 – 60
DRILLING EQUIPMENT PERFORMANCE:											
WITH PNEUMATIC ROCK DRILL (m/h)	60 – 75	15 – 25	20 – 40	60 – 70	8 – 12	10 – 15	200 – 240	50 – 60	100 – 120	50	50
WITH HYDRAULIC ROCK DRILL (m/h)	90 – 110	(25 – 35)	na	90 – 110	na	na	240 – 300	na	120 – 180	na	na
DRILLING-BLASTING PRODUCTION (m³/DRILLED METRE)	1.5 – 2.0	3.0 – 4.0	0.9 – 1.2	1.0 – 1.4	0.7 – 0.9	0.7 – 0.9	1.8 – 2.3	1.5 – 2.5	1.5 – 2.5	8 – 10	14 – 18

Fig. 1.3. Drilling methods for underground operations (Atlas Copco).

1.4 CLASSIFICATION OF THE ROCKS AND THEIR PRINCIPAL PHYSICAL PROPERTIES

Blasthole drilling is almost always carried out in rock masses. For this reason it is important, before starting an operation, to know the different types of materials and their basic properties.

The rock characteristics depend largely upon their origin, divided into three large groups, which will be subject to consideration in the following paragraphs.

1.4.1 *Rock classification according to their origin*

1.4.1.1 *Igneous rocks*

Igneous rocks are formed by solidification from a molten mass, called magma, which is a mixture of dissolved rock and gas material. If the rock has cooled in contact with air or water from the earth's surface it is classified as extrusive or volcanic igneous rock. When the magma cools underneath the surface of the earth it forms intrusive or plutonic igneous rock.

The rate of cooling of the magma gives the grain size of the crystallized minerals; large if it is slow, and small if it is rapid. In the first instance a rock called pegmatite is formed, and in the second it is called aplite. An intermediate case is denominated porpyhry, in which large crystals are observed within a fine grained mass or matrix. These three types are usually found in dike form with strengths that go from one to dozens of meters. The most normal is that of a moderate cooling time which produces a rock mass with a medium size grain, from 1 to 5 mm.

The composition of magma varies during its cooling process because fractioned cristallization is produced depending upon the pressure and temperature of each moment. Also, the residual liquid can react with the minerals that are already solidified and change their chemical content and, again, the original chemical composition of the magmas can have been very different.

The diverse physical and chemical conditions that exist during solidification of magma give the large variety of igneous rocks. They are formed by different minerals, of various sizes and are grouped in diverse ways, the result being that their physical and chemical characteristics are very heterogeneous. Therefore, their behavior when confronted with fragmentation, cutting, wear or meteorization can be varied; although igneous rocks that have not been meteorized, as far as drilling goes, are all hard and compact.

If the rock has a SiO_2 content that is higher than 62%, it is geochemically denominated acid, between that value and and 52%, it is intermediate, between 45 and 52%, basic, and lastly, when lower than 45% it is ultrabasic. In the same sense that igneous rocks are poorer in silica, they are richer in ferromagnesian silicates. The acids are more abrasive and harder than the basic ones, but they are also more dense and resistant to impact.

1.4.1.2 *Metamorphic rocks*

Metamorphic rocks are derived from other pre-existing endogenic or exogenic rocks through important transformations of their mineral components. These marked changes are produced by the necessity of stabilizing their minerals under the new conditions of temperature, pressure and chemism.

These rocks are intermediate in physical and chemical characteristics, between the igneous and the sedimentary, because they have associations of minerals that pertain to the two types. Thus, minerals such as quartz, feldspars, micas, amphiboles, and olivines, essential in igneous rocks, are also found in metamorphic rocks; however they do not contain alkali feldspars. As in sedimentary rocks, they can have calcite, dolomite, silica and hematites; but they do not contain evaporites. Minerals common to the two other types also appear such as tourmaline, zircon, magnetite, topaz and corundum; all of which are very stable in any exogenous or endogenous medium.

There is a series of minerals that are very specific to metamorphic rocks, which can form part of the grains of detrital rocks, owing to their stability in exogenous mediums, and others are at the same time products of meteoric alteration of the minerals in endogenic rocks. Actually, meteorization is a mineralogical transformation that is both a physical and chemical process, but at low temperature and pressure.

1.4.1.3 *Sedimentary rocks*

Sedimentary rocks are formed by accumulation of broken and decomposed rock material, by chemical precipitation of solubilized minerals or by accumulation of shells or other organic material: animal or vegetable.

In the first case, detritic sediments are produced such as gravels, conglomerates or sands in which gravity has played a role in their precipitation. In the second case one can find, as an example, the evaporites or saline rocks precipitated by over-saturation of a brine that is subjected to intense evaporation. The third type are accumulations of shells, skeletons of animals or remains of plants, such as the conchiferous limestones. This last group is subdivided into organogenous biochemistry and mineral biochemistry depending upon whether their components are of organic or inorganic nature. For the first we have coal and petroleum, and for the second the limestones, dolomites and phosphatic rocks.

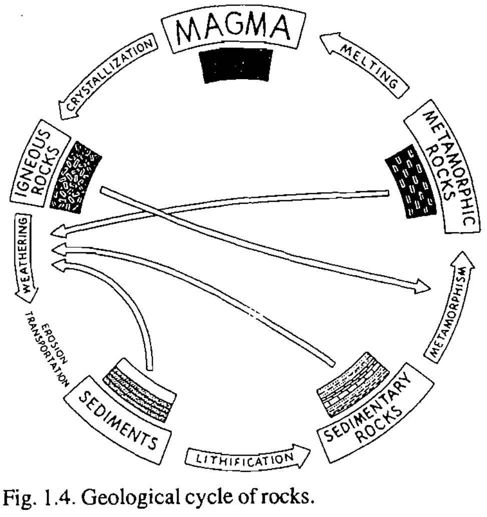


Fig. 1.4. Geological cycle of rocks.

Table 1.1

Classification	Mohs' scale of hardness	Compressive strength (MPa)
Very hard	+7	+200
Hard	6-7	120-200
Medium hard	4.5-6	60-120
Medium soft	3-4.5	30-60
Soft	2-3	10-30
Very soft	1-2	−10

For an initial classification of sedimentary rocks, their formation process is taken into account, later the grain size, the characteristics of their bonding, apart from the types and quantities of their mineral components.

1.4.2 *Rock properties that affect drilling*

The principal physical rock properties that have influence upon penetration mechanisms and, as a consequence, on choice of the drilling method are: hardness, strength, elasticity, plasticity, abrasiveness, texture, structure, characteristics of breakage.

1.4.2.1 *Hardness*

Hardness is considered to be the resistance of a surface layer to be penetrated by another body of harder consistency.

In rock, it is a function of the hardness and composition of its mineral grains, the porosity, degree of humidity, etc.

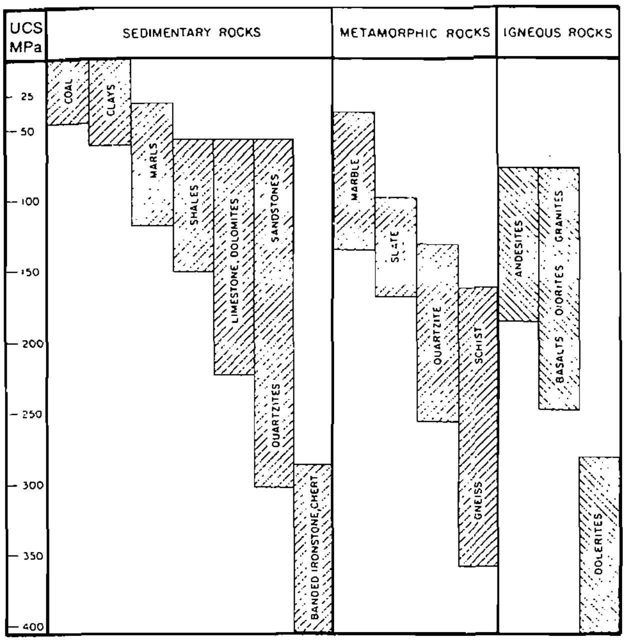
The hardness of rocks is the principal type of resistance that must be overcome during drilling, because once the bit has penetrated, the rest of the operation is easier.

Rocks are classified as to their hardness by using Friedrich von Mohs' Scale of Hardness (1882), in which the concept is that any mineral can scratch anything that has a lower or equal number to it, numbering from 1 to 10. As can be seen from Table 1.1, there is a certain correlation between hardness and compressive strength of the rocks.

1.4.2.2 *Strength*

Mechanical strength of a rock is the property of opposing destruction by an external force, either static or dynamic.

The rocks give maximum resistance to compression, normally, as the tensile strength is not more than 10 or 15% of the compressive strength. This is due to the fragility of rocks, to the large quantity of local defects and irregularities that exist and to the small cohesion between the particles of which they are constituted.



NOTE UCS : Uniaxial Compressive Strength

Fig. 1.5. Most frequent compressive strengths for different types of rocks.

The rock strength fundamentally depends on its mineralogical composition. Among the integrating minerals, quartz is the most solid with a strength that goes over 500 MPa, while that of the ferromagnesian silicates and the aluminosilicates vary between 200 and 500 MPa, and that of calcite from 10 to 20 MPa. Therefore, the higher the quartz content, the more the strength increases.

The mineral strength depends upon the size of the crystals and diminishes with their increase. This influence is significative when the crystal size is under 0.5 mm.

In rocks, the size factor has less influence on strength as the intercrystalline cohesion force also intervenes. For example, the compressive strength of a fine grained arkose sandstone is almost double that of a coarse grained; that of marble composed of 1 mm graines is equal to 100 MPa, whereas a fine grained limestone – 3 to 4 mm – has a strength of 200 to 250 MPa.

Amongst the sedimentary rocks the ones with highest strength are those that contain silica cement. With the presence of clay cement, the strength is drastically reduced.

Porosity in rocks with the same lithology also reduces strength proportionately, more porosity – less strength; as it simultaneously reduces the number of contacts of the mineral particles and the force of reciprocal action between them.

The depth at which rocks were formed and the degree of metamorphism also have influence upon their strength. Therefore, the strength of clay beddings near the ground surface can be of 2 to 10 MPa, whereas in clay rocks that went through a certain metamorphism the strengths can reach 50 to 100 MPa.

On the other hand, the strength of ansiotropic rocks depends upon the sense of action of the force. The compressive strength of rocks in the perpendicular to stratification sense or schistosity is larger than in a parallel sense. The quotient that is usually obtained between both strength values varies between 0.3 and 0.8, and it is equal to 1 only for isotropic rocks.

In Fig. 1.5, the most frequent compressive strengths for different types of rock are indicated.

1.4.2.3 *Elasticity*

The majority of rock minerals have an elastic-fragile behavior, which obeys the Law of Hooke, and are destroyed when the strains exceed the limit of elasticity.

Depending upon the nature of deformation, as function of the stresses produced by static charges, three groups of rocks are taken into consideration: 1) The elastic-fragile or those which obey the Law of Hooke, 2) The plastic-fragile, that have plastic deformation before destruction, 3) The highly plastic or very porous, in which the elastic deformation is insignificant.

The elastic properties of rocks are characterized by the elasticity module 'E' and the Poisson coefficient 'ν'. The elasticity module is the proportionality factor between the normal stress in the rock and the relative correspondant deformation, its value in most rocks varies between 0.03 × 10^4 and 1.7 × 10^5 MPa, basically depending upon the mineralogical composition, porosity, type of deformation and magnitud of the applied force.

The values of the elasticity modules in the majority of sedimentary rocks are lower than those corresponding to the minerals in their composition. The texture of the rock also has influence on this parameter, as the elasticity module in the direction of the bedding or schistosity is usually larger than when perpendicular.

Poisson's coefficient is the factor of proportionality between the relative longitudinal deformations and the transversal deformations. For most rocks and minerals it is between 0.2 and 0.4, and only in quartz is it abnormally low, around 0.07.

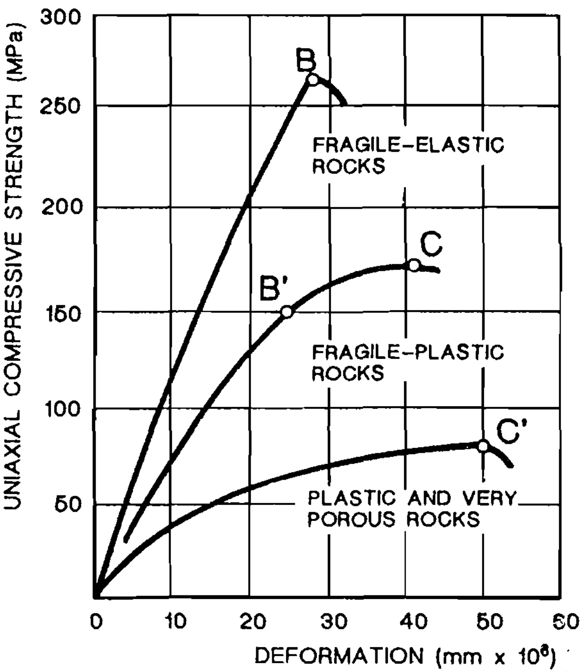


Fig. 1.6. Curves of stress-deformation for different types of rocks.

1.4.2.4 *Plasticity*

As indicated before, in some rocks the plastic deformation preceeds destruction. This begins when the stresses exceed the limit of elasticity. In the case of an ideally plastic body, that deformation is developed with an invariable stress. Real rocks are deformed and consolidated at the same time: in order to increase the plastic deformation it is necessary to increase the effort.

The plasticity depends upon the mineral composition of the rocks and diminishes with an increase in quartz content, feldspar and other hard minerals. The humid clays and some homogeneous rocks have plastic properties.

The plasticity of the stony rocks (granites, schistoses, crystallines and sandstones) becomes noticeable especially at high temperatures.

1.4.2.5 *Abrasiveness*

Abrasiveness is the capacity of the rocks to wear away the contact surface of another body that is harder, in the rubbing or abrasive process during movement.

The factors that enhance abrasive capacities of rocks are the following:

– The hardness of the grains of the rock. The rocks that contain quartz grains are highly abrasive.

– The shape of the grains. Those that are angular are more abrasive than the round ones.

– The size of the grains.

– The porosity of the rock. It gives rough contact surfaces with local stress concentrations.

– The heterogeneity. Polymineral rocks, although these are equally hard, are more abrasive because they leave rough surfaces with hard grains as, for example, quartz grains in a granite.

This property has great influence upon the life of drill steel and bits.

In Table 1.2, the mean amounts of quartz for different types of rock are indicated.

Table 1.2

Rock type	Quartz content %
Amphibolite	0-5
Anorthosite	0
Diabase	0-5
Diorite	10-20
Gabbro	0
Gneiss	15-50
Granite	20-35
Greywacke	10-25
Limestone	0-5
Marble	0
Mica gneiss	0-30
Mica schist	15-35
Norite	0
Pegmatite	15-30
Phyllite	10-25
Quartzite	60-100
Sandstone	25-90
Shale	0-20
Slate	10-35
Taconite	0-10

1.4.2.6 *Texture*

The texture of a rock refers to the structure of the grains of minerals that constitute it. The size of the grains are an indication, as well as their shape, porosity etc. All these aspects have significative influence on drilling performance.

When the grains have a lenticular shape, as in a schist, drilling is more difficult than when they are round, as in a sandstone.

The type of material that makes up the rock matrix and unites the mineral grains also has an important influence.

As to porosity, those rocks that have low density and, consequently, are more porous, have low crushing strength and are easier to drill.

In Table 1.3 the classification of some types of rocks is shown, with their silica content and grain size.

Table 1.3. Common rock names and their geological definitions (based on Dearman, 1974; ISRM, 1981a).

<table>
<tr><td colspan="2">Genetic Group</td><td colspan="9">Sedimentary</td><td colspan="2">Metamorphic</td><td colspan="3">Igneous</td></tr>
<tr><td colspan="2" rowspan="2">Structure</td><td colspan="9">Bedded</td><td>Foliated</td><td colspan="4">Massive—Jointed</td></tr>
<tr><td colspan="8">Fragmental (detrital grains)</td><td colspan="6">Crystalline or glassy (cryptocrystalline)</td></tr>
<tr><td rowspan="2">Grain size, mm</td><td rowspan="2">Texture</td><td colspan="2" rowspan="2">Grains are of rock, quartz, feldspar, and clay minerals</td><td colspan="2" rowspan="2">50% grains are of carbonate</td><td colspan="2" rowspan="2">50% grains are of fine-grained igneous rock</td><td rowspan="2" colspan="2">Chemical-organic rocks</td><td rowspan="2">Quartz, feldspars, micas, acicular dark minerals</td><td rowspan="2">Depends on parent rock</td><td colspan="3">Light-colored minerals are quartz, feldspar, mica, and feldsparlike minerals</td></tr>
<tr><td>Acid</td><td>Intermediate</td><td>Basic</td></tr>
<tr><td></td><td>Very coarse-grained</td><td rowspan="2">Rudaceous</td><td rowspan="2">Grains are of rock fragments
Rounded grains: conglomerate
Angular; grains: breccia</td><td rowspan="5">Limestone</td><td rowspan="2">Calcirudite</td><td rowspan="5">Volcanic ash</td><td rowspan="2">Rounded grains: agglomerate
Angular grains: volcanic breccia</td><td rowspan="5" colspan="2">Saline rocks: halite, anhydrite, gypsum,
Limestone
Dolomite
Peat
Lignite
Coal</td><td rowspan="5">Gneiss: alternate bands of granular and flaky minerals</td><td rowspan="5">Quartzite
Marble
Granulite
Hornfels
Amphibolite</td><td>Pegmatite</td><td></td><td></td></tr>
<tr><td>60</td><td>Coarse-grained</td><td>Granite</td><td>Diorite</td><td>Gabbro</td></tr>
<tr><td>2</td><td>Medium-grained</td><td>Arenaceous</td><td>Sandstone: grains are mainly mineral fragements
Quartz sandstone: 95% quartz, voids empty or cemented
Arkose: 75% quartz, up to 23% feldspar, voids empty or cemented
Graywacke: 73% quartz, 15% fine detrital matrix, rock and feldspar fragments</td><td>Calcarenite</td><td rowspan="3">Tuff</td><td>Microgranite</td><td>Microdiorite</td><td>Dolerite</td></tr>
<tr><td>0.06</td><td>Fine-grained</td><td rowspan="2">Argillaceous or lutaceous</td><td rowspan="2">Mudstone
Shale: fissile mudstone
Siltstone: 50% fine-grained particles –
Claystone: 50% very fine-grained particles</td><td rowspan="2">Calcilutite (chalk)</td><td rowspan="2">Rhyolite</td><td rowspan="2">Andesite</td><td rowspan="2">Basalt</td></tr>
<tr><td>0.002</td><td>Very fine-grained</td></tr>
<tr><td></td><td>Glassy</td><td colspan="6"></td><td colspan="2">Chert
flint</td><td colspan="2"></td><td colspan="3">Volcanic glasses: obsidian, pitchstone, tachylite</td></tr>
</table>

Table 1.4. Properties of rock types according to origin-based classification.

Rock type		Specific gravity (m^3)	Grain size (mm)	Swell factor	Compressive strength (MPa*)
Intrusive	Diorite	2.65-2.85	1.5-3	1.5	170-300
	Gabbro	2.85-3.2	2	1.6	260-350
INGENOUS	Granite	2.7	0.1-2	1.6	200-350
	Andesite	2.7	0.1	1.6	300-400
Extrusive	Basalt	2.8	0.1	1.5	250-400
	Rhyolite	2.7	0.1	1.5	120
	Trachyte	2.7	0.1	1.5	330
	Congomerate	2.6	2	1.5	140
	Sandstone	2.5	0.1-1	1.5	160-255
SEDIMENTARY	Shale	2.7	1	1.35	70
	Dolomite	2.7	1-2	1.6	150
	Limestone	2.6	1-2	1.55	120
	Limerock	1.5-2.6	1-2	1-1.6	30-100
	Gneiss	2.7	2	1.5	140-300
	Marble	2.7	0.1-2	1.6	100-200
METAMORPHIC	Quartzite	2.7	0.1-1	1.55	160-220
	Schist	2.7	0.1-1	1.6	60-400
	Serpentine	2.6	–	1.4	30-150
	Slate	2.7	0.1	1.5	150

* $1\ \text{MPa} = 1\ \text{MN/m}^2 = 10\ \text{kg/cm}^2 = 142.2\ \text{psi}$

In Table 1.4, the characteristic properties of different types of rocks are indicated, according to their origin.

1.4.2.7 *Structure*

The structural properties of the rock masses, such as schistosity, bedding planes, joints, diabases and faults, as well as their dip and strike affect the allignment of the blastholes, the drilling performance and the stability of the blasthole walls.

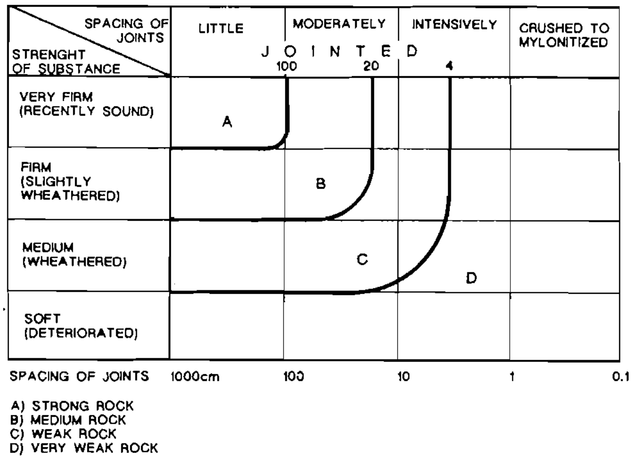


Fig. 1.7. Classification of the rock masses.

In Fig. 1.7, the rock masses are classified from the spacing between joints and the strength of the rocky material.

REFERENCES

Atlas Copco: *Manual atlas copco,* 4th edition. 1984.

Heinz, W.F.: *Diamond drilling handbook.* 1989.

Hunt, R.E.: *Geotechnical engineering techniques and practices.* McGraw Hill. 1986.

Sandvik-Coromant: *Manual de perforación de rocas. Teoría y técnica.* 1983.

Tamrock: *Handbook of surface drilling.* 1989.

CHAPTER 2

Rotary percussive drilling

2.1 INTRODUCTION

Drilling by rotary percussion is the most classic system for drilling blastholes, and its chronological appearance coincides with the industrial development of the nineteenth century. The first prototype machines made by Singer (1838) and Couch (1848) were run by steam, but it was when compressed air was used as the source of energy, in the execution of the tunnel of Mont Cenis in 1861, that this system evolved and was put into extensive use. This event, along with the arrival of dynamite, was decisive in the rapid development of rock breakage in mining and civil engineering at the end of the last century.

The drilling principle of these rigs is based upon the impact of a steel piece (piston) that hits a utensil which transmits at the same time that energy to the bottom of the blasthole by means of the final element called the bit. The rotary percussive rigs are classified in two large groups, depending upon where the hammer is located:

– *Top hammer.* In these drills, two of the basic actions, rotation and percussion, are produced outside the blasthole, and are transmitted by the shank adaptor and the drill steel to the drill bit. The hammers can be driven hydraulically or pneumatically.

– *Down the hole hammer.* The percussion is delivered directly to the drill bit, whereas the rotation is performed outside the hole. The piston is driven pneumatically, while the rotation can be hydraulic or pneumatic.

Depending upon the fields of application of these drilling rigs, surface or underground, the most common range of diameters are shown in Table 2.1.

The main advantages of rotary percussive drilling are:

– It can be applied to any type of rock, from soft to hard.

– Wide range of diameters;

– Versatile equipment, it adapts well to different operations and is very mobile;

– Only requires one operator;

– Easy, quick maintenance, and

– The capital cost is not high.

In view of these advantages and characteristics, the type of operations where it is used are:

– Underground civil engineering; tunnels, underground hydraulic plants, residual deposits, etc., and in surface operations; roads, highways, industrial excavations, etc.

– In underground mines and in small to medium sized surface operations.

2.2 FUNDAMENTALS OF ROTARY PERCUSSIVE DRILLING

Rotary percussion drilling is based upon the combination of the following:

– *Percussion.* The impacts produced by repeated blows of the piston generate shock waves that are transmitted to the bit through the drill steel (in top hammer) or directly upon it (down the hole).

– *Rotation.* With this movement, the bit is turned so that the impacts are produced on the rock in different positions.

– *Feed, or thrust load.* In order to maintain the contact of the drill bit with the rock, a thrust load or feed force is applied to the drill string.

– *Flushing.* Flushing removes the drill cuttings from the blasthole.

The indentation forming process with which penetration is achieved in this drilling system is divided into five times, as indicated in Fig. 2.2.

a) Crushing of the rough edges of the rock upon bit contact.

b) Radial cracks appear from the points of stress concentration and a V shaped wedge is formed.

c) The rock of the wedge is pulverized.

d) The larger fragments are chipped in the zones next to the wedge.

e) The drill cuttings are flushed away.

This sequence repeats itself with the same impact rhythm of the piston upon the system of energy transmission to the bit.

The yield of this process increases proportionally with the size of the rock chippings.

2.2.1 *Percussion*

The kinetic energy E_c of the piston is transmitted from the hammer to the drill bit, through the drill steel, in the form of a shock wave. The wave travels at high speed and its shape depends basically on the design of the piston.

When the shock wave reaches the drill bit, part of the energy is transformed into work, causing the bit to penetrate, and the rest is reflected and returns through the drill steel. The efficiency of this transmission is difficult to

Table 2.1.

Drilling method	Drilling diameter (mm)	
	Surface	Underground
Top hammer	50 – 127	38 – 65
Down the hole	75 – 200	100 – 165

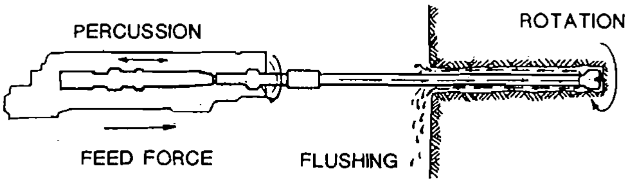


Fig. 2.1. Basic actions in rotary percussive drilling.

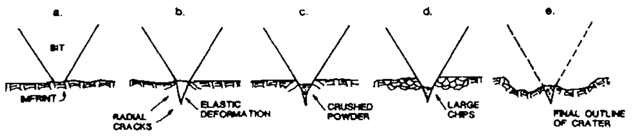


Fig. 2.2. Sequence of rock failure during center formation (Hartman, 1959).

evaluate as it depends upon many factors such as: type of rock, shape and size of piston, drill steel characteristics, bit design, etc. Another thing to take into account is that energy is lost through the sleeves of the rod couplings, due to reflection and fricton which is converted into heat and wear on the drill steel threads. In the first coupling the losses oscillate between 8 and 10% of the shock wave energy.

In down the hole drilling the piston energy is transmitted directly to the bit, giving greater performance.

In these drilling systems, percussion force is the parameter that most influences the penetration rate.

The energy freed per hammer stroke can be estimated from the following equations:

$$E_c = \tfrac{1}{2}\, m_p \times v_p^2$$

$$E_c = p_m \times A_p \times l_p$$

where: m_p = Mass of the piston, v_p = Maximum piston speed, p_m = Pressure of the work fluid (oil or air) inside the cylinder, A_p = Surface area of the piston face, l_p = Stroke of the piston.

In the majority of hydraulic hammers, the manufacturers indicate the impact energy value, but this is not the case with the pneumatic hammers. Special care should be taken in estimating the p_m for these, as it is 30 to 40% lower in the cylinder than in the compressor, owing to charging and expansion losses of air with each stroke of the piston.

Thus, the hammer power is the energy per stroke multiplied by the frequency of strokes n_g:

$$P_M = E_c \times n_g \text{ where } n_g = K \times \left(\frac{p_m \times A_p}{m_p \times l_p}\right)^{1/2}$$

and taking into account the previous equations, the following can be stated:

$$P_M = k \times \frac{(p_m \times A_p)^{3/2} \times l_p^{1/2}}{m_p^{1/2}}$$

The percussion mechanism consumes from 80 to 85% of the total power of the equipment.

2.2.2 *Rotation*

Rotation, which turns the drill bit between consecutive blows, has the function of making the bit strike upon different points of the rock in the bottom of the blasthole. In each type of rock there is an optimum rotation speed which produces larger sized cuttings taking advantage of the free area of the hole created with each impact.

When drilling with insert bits, the most common rotation speeds oscillate between 80 and 150 r.p.m. with angles between indentations of 10 to 20°, Fig. 2.3. For button bits from 51 to 89 mm, the speeds should be lower, between 40 and 60 r.p.m., that bring turning angles between 5 and 7°. Bits of larger diameters require even lower speeds.

2.2.3 *Thrust load*

The energy generated by the mechanism of hammer blows should be transfered to the rock, for which it is necessary to have the drill bit in permanent contact with the bottom of the hole. This is achieved with the thrust load or pull down, supplied by a pull down motor, which should be adapted to rock type and drill bit.

Insufficient thrust load has the following negative effects: lower penetration rates, greater wear of rods and sleeves, loosening of drill steel threads and heating of the same. On the contrary, if the pull down is excessive the penetration rate is also diminished, there is increased

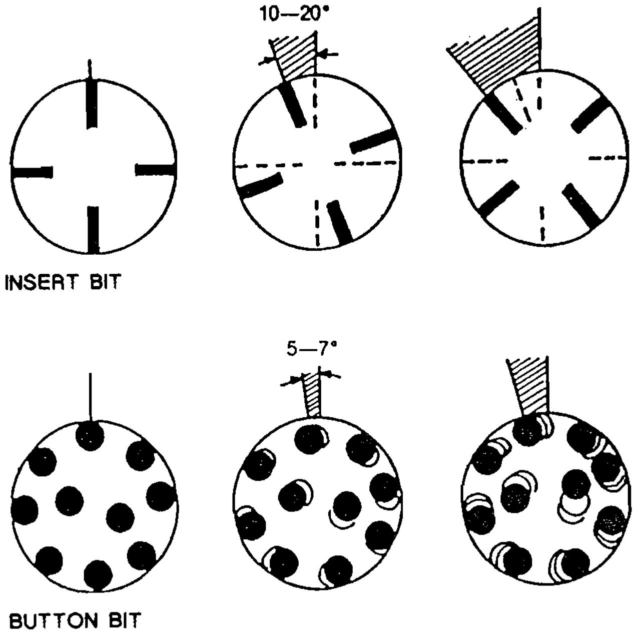


Fig. 2.3. Rotation speed between consecutive blows as a function of penetration rate and bit diameter.

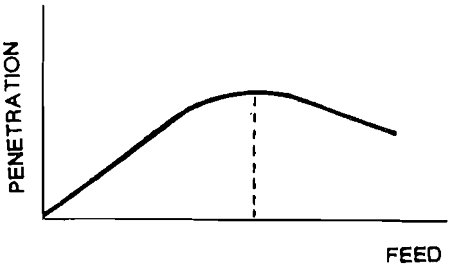


Fig. 2.4. The effect of thrust load upon penetration rate in top hammer drilling.

rotation resistance, drill steel can become jammed, the wear on the bits increases as well as the rotation rate and equipment vibrations, and the blastholes can be deviated.

As occurs with rotation, this parameter does not have decisive influence on the penetration rates, Fig. 2.4.

2.2.4 *Flushing*

In order to have efficient drilling, the bottoms of the blastholes must be maintained clean by evacuating drill cuttings as soon as they appear. If this is not done, a large quantity of energy will be consumed in regrinding with the consequent wear on drill bits and decrease in penetration, apart from the risk of jamming.

Blasthole flushing is carried out with a flow of air, water or foam that is injected by pressure to the bottom through an opening in the center of the drill steel and flushing holes in the drill bits.

The cuttings are removed up through the space between the rod and the blasthole walls, Fig. 2.5.

Flushing with air is used in surface operations, where the dust produced can be eliminated by means of dust collectors.

Water flushing is mostly used in underground drilling, which also keeps dust down, although it reduces performance by about 10 to 20%.

Foam is used as a complement to air as it helps bring large particles up to the surface and also acts as a sealer for blasthole walls when drilling through loose material.

The velocity of air flow for efficient cleaning with air goes from 15 to 30 m/s. The minimum velocities for each case can be calculated from the following equation:

$$v_a = 9{,}55 \times \frac{\rho_r}{\rho_r + 1} \times d_p^{0.6}$$

where: v_a = Velocity of air flow (m/s), ρ_r = Rock density (g/cm^3), d_p = Diameter of the particles (mm).

Therefore, the flow that should be supplied by the compressor is:

$$Q_a = \frac{v_a \times (D^2 - d^2)}{1.27}$$

where: Q_a = Flow (m^3/min), D = Blasthole diameter, d = Diameter of the rods (m).

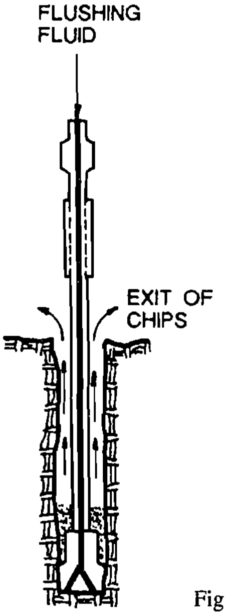


Fig. 2.5. Principle of flushing.

When water is used for flushing, the velocity of air flow should be between 0.4 and 1 m/s. In these cases the pressures are maintained between 0.7 and 1 MPa, to keep the flow from entering into the hammer.

When using air with top hammers, it is not common to have a high pressure compressor for flushing alone. Only in down the hole hammer drilling is a high pressure compressor used (1 – 7 MPa) because the percussion power is increased along with the flushing of cuttings.

An important factor to remember when estimating the flushing flow is that of charging losses produced due to the narrow conducts through which the fluid must pass (flushing needle, drill steel holes) as well as along the drill string.

In Table 2.2, the flushing velocities for top hammer drilling are indicated as function of air compressor flow and drill steel diameter.

2.3 TOP HAMMER DRILLING

This drilling system can be qualified as the most conventional or classic, and although its use by pneumatic drive was limited by the down the hole and rotary equipment, the appearance of the hydraulic hammers in the sixties has given a new boost to this method, complementing and widening its field of application.

2.3.1 *Pneumatic drilling rigs*

Hammers driven by compressed air basically consist in:

– A cylinder with a front cover that has an axial opening where the rotation chuck goes, as well as a retaining device for the drill rods.

– The piston that alternately strikes the drill steel shank through which the shock wave is transmitted to the rod.

– The valve that regulates the passage of compressed air in a pre-set volume and in alternating form to the front and back of the piston.

– A rotation mechanism, that can be a spirally fluted rifle bar or of independent rotation.

– A flushing system that consists in a tube that allows the passage of air to the inside of the drill steel.

These elements are common to all the types of hammers on the market, with only a few design characteristics that differ: diameter of the cylinder, length of the piston stroke, distribution valves, etc. The following describes the working principle of a pneumatic hammer, Figs. 2.6 to 2.12.

1. The piston is at the end of its return stroke and is ready to start its working stroke. The air, at line pressure, fills the backhead (1) and passes through the back supply port (2) into the cylinder (3). The air pushes the piston forward, beginning the working stroke. Meanwhile, the cylinder front end (5) is at atmospheric pressure since the exhaust port (6) is open.

2. The piston (4) continues to accelerate forward, driven by the line pressure, until the leading edge (7) of the pistons control head shuts off the entrance of compressed air. The air confined in the back end of the cylinder (3) starts to expand and contiunes to drive the piston forward. Note that the piston flange (4) closes the exhaust port (6) and that the front end is still at atmospheric pressure.

3. The air confined at the back of the piston (3) continues to expand until the back edge of the piston flange starts to uncover the exhaust port (6). Remember that the piston control head (7) has already shut off the compressed air entrance, so that no compressed air will be wasted when the exhaust port is opened.

Up front, the piston has trapped air that was a atmopheric pressure (5), and has now compressed it to slightly above atmospheric pressure.

4. The piston continues to move forward because of its momentum until it strikes the drill shank steel. Now, the back edge of the piston flange (8) has uncovered the exhaust port (6) and the air in the back end is exhausted into the atmosphere. While this was going on, the back edge (10) of the control head opened the front supply port admitting compressed air to the front end (5) driving the piston back on the return stroke. During this stage there is compressed air pushing against the piston from the front end (5) and also pushing against the back end (10). The front surface area is much larger than the back (10) so the piston moves towards the rear.

5. The piston is accelerated back on the return stroke, until the back edge of the control head (10) covers up the front air supply port. The air up front then continues to push the piston back.

6. The piston continues to accelerate backwards while the air in the front end (5) expands until the front end of the piston flange (11) uncovers the exhaust port, trapping the air in the back end of the cylinder and compressing it to a pressure slightly more than atmospheric. Note than the front edge of the control head (7) is just about to open the back supply port.

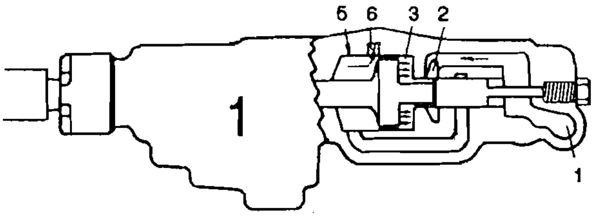


Fig. 2.6. Piston at the end of its return stroke.

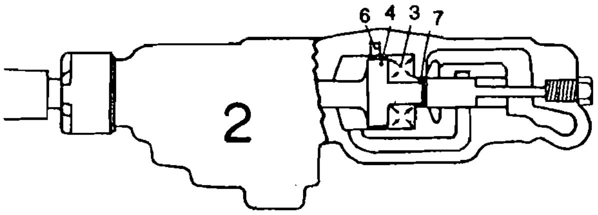


Fig. 2.7. The piston accelerates forward.2.

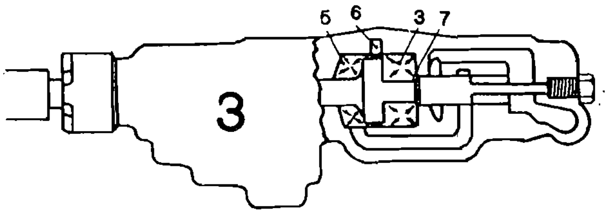


Fig. 2.8. The back edge of the piston flange uncovers the exhaust port.

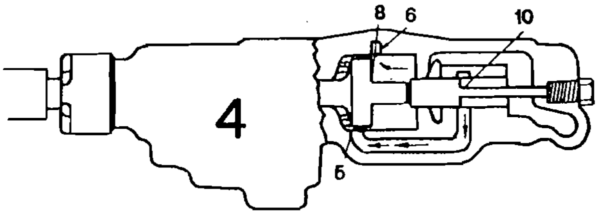


Fig. 2.9. The piston compresses the air in front of it.

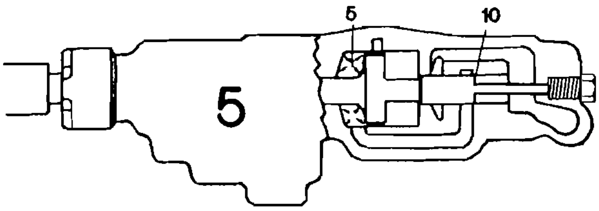


Fig. 2.10. The piston is accelerated back.

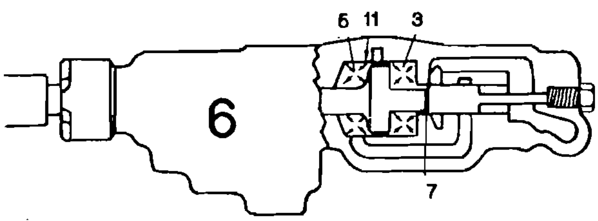


Fig. 2.11. The front edge of the piston flange uncovers the exhaust port.

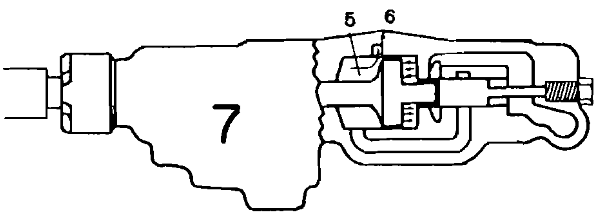


Fig. 2.12. Return stroke of the piston finishes.

Table 2.2.

Flow (m^3/min)	3.2	5.2	6.5	6.5	9.3	9.3	9.3	9.3
Rod diameter (mm)	32	38	38	45	45	51	87	100
Coupling diameter (mm)	45	55	55	61	61	72	–	–
Flushing hole diameter (mm)	12	14	14	17	17	21	61	76
Blasthole diameter	Flushing air velocity (m/s)							
51 mm (2″)	42	–	–	–	–	–	–	–
64 mm (2½″)	22	42	52	–	–	–	–	–
76 mm (3″)	15	25	32	37	50	–	–	–
89 mm (3½″)	–	17	21	24	27	36	–	–
102 mm (4″)	–	–	15	17	22	24	68	–
115 mm (4½″)	–	–	–	12	17	18	34	69
127 mm (5″)	–	–	–	–	13	15	19	34
140 mm (5½″)	–	–	–	–	–	–	16	21
152 mm (6″)	–	–	–	–	–	–	–	15

Table 2.3. Mean characteristics of pneumatic hammers.

Relationship piston diameter/Blasthole diameter	1.5-1.7
Piston stroke (mm)	35-95
Frequency of Blows (blows/min)	1500-3400
Rotation velocity (r/min)	40-400
Relative air consumption (m^3/min. cm. diameter)	2.1-2.8

7. The return stroke finishes when the back supply port is fully open, allowing compressed air to enter behind the piston. This has a cushioning effect, bringing the piston to a gentle stop, and at the same time prepares the piston for a new working stroke.

Some of the typical characteristics of this equipment are indicated in Table 2.3.

The drilling lengths reached with this system do not usually go over 30 m, owing to important energy losses in transmission of the shock wave and blasthole deviations.

As mentioned before, drill steel rotation can be obtained by two different procedures:

a) With spirally fluted rifle bar or ratchet ring, and

b) With an independent motor.

The first group is common in light drilling rigs, whereas the second is used for large diameter blastholes where higher performance is necessary.

In rotation by fluted rifle bar, the piston has a tubular shape and surrounds the bar by means of a rifle nut. The bar is connected to the static components of the hammer with rachets, Fig. 2.13. The front end of the piston has splines that fit into those of the rotation chuck. This makes the piston, in its return stroke, turn in the same direction as the drill steel. The rifle bars come in different pitches, so that every 30, 40 or 50 strokes a complete turn is obtained.

In rotation by rachet ring, the front end of the piston has splines that register in the chuck bushing, which is internally linked with the rachet ring. Also, in this case, the drill steel only turns during the return stroke of the piston.

The second procedure, which is the most widely used, has a separate motor mounted outside the pneumatic or hydraulic hammer. Its advantages are:

– With a piston of the same size the hammer receives more energy, because when the rifle-bar is eliminated the working surface area of the piston on which the compressed air acts is increased.

– More power which allows working with larger diameters and lengths.

– The percussion and rotation can be adapted to the type of rock to be drilled.

– Increased drilling yield.

This type of drills have cylindrical gearing to transmit the rotation movement to the drill steel, Fig. 2.14.

The field of application of top hammer pneumatic rock drills is turning more and more to short blastholes with lengths between 3 and 15 m, blastholes of small diameter, from 50 to 100 mm, hard rocks and areas of difficult access. This is due to the large consumption of compressed air, about 2.4 m^3/min for each centimeter of diameter, and to the great wear on accessories, drill steel, sleeves, bits, etc., to the frequency of blows and shape of the shock wave transmitted with large diameter pistons.

However, pneumatic rock drills still give numerous advantages:

– Simplicity,

– Reliability and low maintenance,

– Easy repair,

– Low capital cost, and

– Possibility of using the existing compressed air installations for underground operations.

2.3.2 *Hydraulic rock drills*

At the end of the sixties and beginning of the seventies, a great technological advance took place in rock drilling with the development of hydraulic hammers.

A hydraulic drill is composed basically of the same elements as the pneumatic, Fig. 2.15.

The most important difference between both systems is that instead of using compressed air as an energy transmission medium, this has been replaced by a more efficient medium, hydraulic oil, which is pumped around the circuits by gear or piston pumps, driven by a diesel engine or electric powerpack.

The following describes the working principle of a hydraulic top hammer drill used in surface drilling, Figs. 2.16 to 2.19.

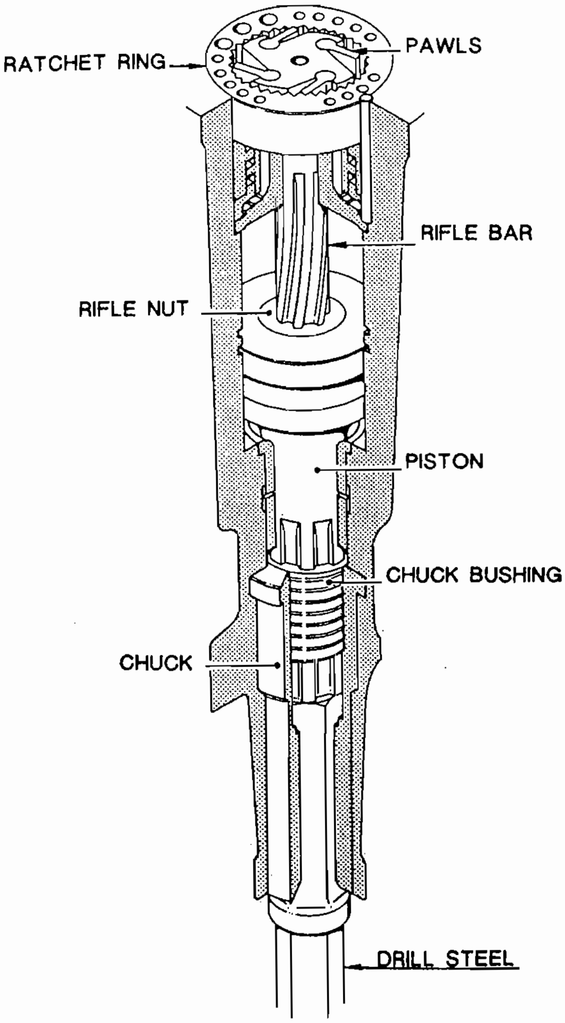


Fig. 2.13. Operational parts for rifle-bar rotation.

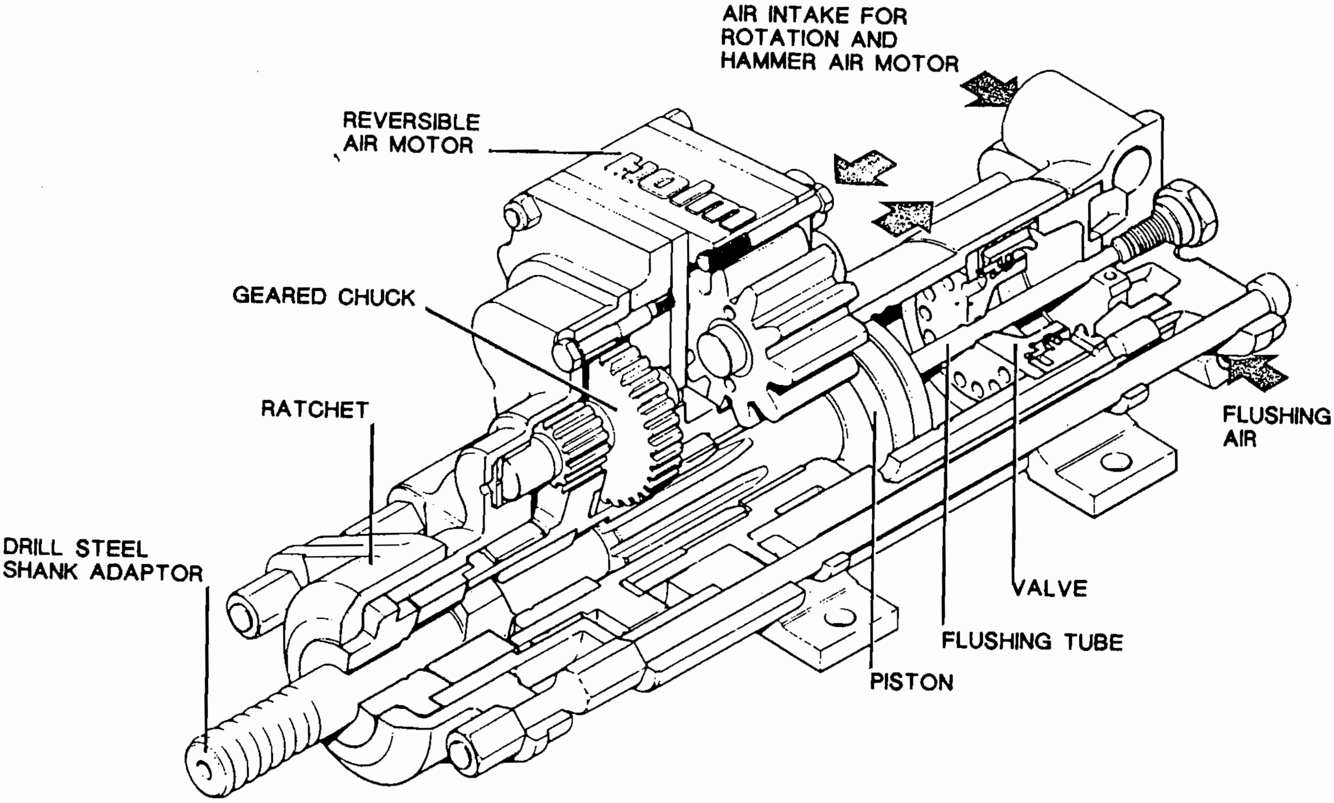


Fig. 2.14. Cross-section of a pneumatic rock drill with independent rotation (Compair-Holman).

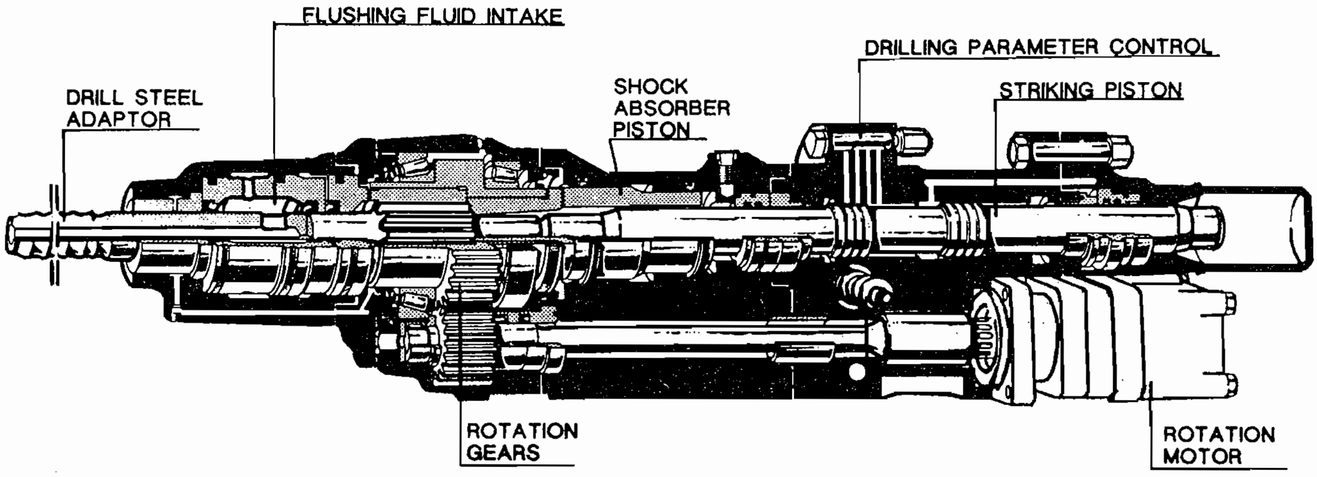


Fig. 2.15. Cross-section of a hydraulic rock drill (Atlas Copco).

1. The piston is shown at the front end of its stroke. The hydraulic oil enters the rock drill through the high pressure port (1) and flows into the front part of the cylinder (2). It forces the piston backwards and at the same time enters the distributor chamber (3) pushing the distributor (4) to the rear position. Part of the oil delivery enters the high pressure accumulator (5) compressing its nitrogen and thus accumulating energy. In this position the oil in the rear part of the cylinder escapes through the port (6) to the return port (7). The low pressure accumulator (8) functions similarly to prevent shock loads in the return hoses.

2. When the piston has travelled so far that the edge (9) has covered the ports (6), port (10) will have opened and pressure acting on the high pressure side stops the piston. Pressure shock caused by the piston is absorbed in the accumulator (5). After this, edge (11) uncovers ports (12) and the pressurized oil in the distributor chamber escapes to the return line. Before this, edge (13) prevents oil flow to the distributor chamber, and the pressure in the front part of the cylinder forces the piston backwards.

3. As the pressure in the distributor is reduced, the high pressure prevailing against the rear face of the distributor (4) forces it forward, thus covering the escape ports (6). In this position the oil can flow to the rear part of the cylinder through a pressure port (14) between the distributor and the body. At the same time the oil can flow through port (10) to the cylinder.

4. The piston moves forward due to the imbalance of forces prevailing in the front and rear parts of the cylinder. At the same time the HP-accumulator (5) discharges oil to the HP line thus increasing the oil flow to the cylinder. Shortly before percussion of the piston, edge (12) allows oil to flow towards the distributor chamber and the imbalance of forces between distributor faces moves the distributor to the rear position, closing the oil supply to the rear part of the cylinder. After the instant of percussion, the return cycle of the piston begins as described before.

Although in the beginning these drills were mostly used in underground operations, as time went by they began to appear in surface drilling as a complement to the pneumatic drills.

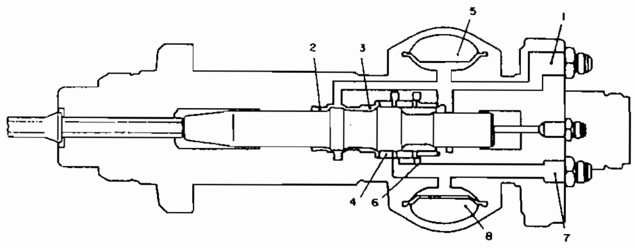


Fig. 2.16. The piston is at the front end of its stroke.

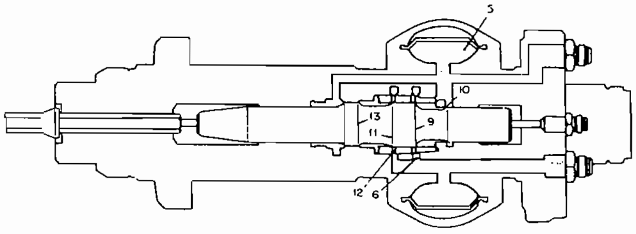


Fig. 2.17. The piston moves backwards.

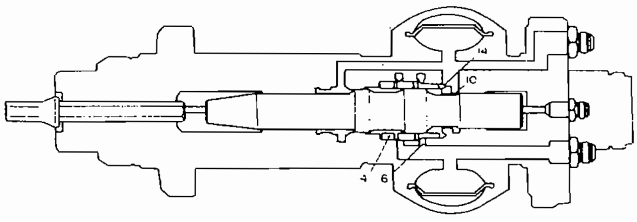


Fig. 2.18. The piston in rear position.

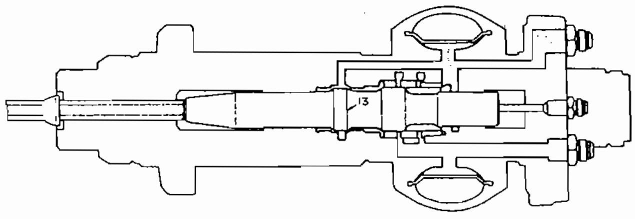


Fig. 2.19. The piston moves forward.

Table 2.4. Mean characteristics of hydraulic top hammer drills.

Work pressure (MPa)	7.5-25
Impact power (kW)	6-20
Frequency of blows (blows/min)	2000-5000
Rotation velocity (r/min)	0-500
Maximum force (Nm)	100-1800
Relative air consumption (m^3/min cm. diameter)	0.6-0.9

Table 2.5.

Diameter of the drill steel (mm-inches)	Output power of the drill (kW)
25.4-1″	8-12
31.7-1¼″	10-14
38.1-1½″	14-16
44.5-1¾″	16-18
50.8-2″	18-22

The characteristics of these drills is shown in Table 2.4. Depending upon the output power of the rock drill, the drill steel size can be chosen. Table 2.5, can be taken as a general guideline.

The reasons for which hydraulic drilling is technologically better than pneumatic are the following:

– *Lower energy consumption.* Hydraulic drills work with fluids at much higher pressures than the pneumatically driven and, also, there is not as much pressure fall. Energy produced by hydraulic power is more efficient and costs one-third of that obtained with pneumatic drilling.

– *Lower drilling accessory costs.* Energy transmission in hydraulic rock drills is carried out by means of pistons that are longer and with smaller diameters than those corresponding to pneumatic drills. Drill steel fatigue depends upon its sections and the piston size because, as observed in Fig. 2.20, the shape of the shock wave is much clearer and uniform in hydraulic drills than in pneumatic, where stress levels are very high, producing steel fatigue and a series of low energy secondary waves. In practice it has been proven that the life of drill steel is prolonged by 20% in hydraulic drills.

– *Greater drilling capacity.* Owing to better energy transmission and wave shape, the penetration rates of hydraulic drills are 50 to 100% more than those of pneumatic equipment.

– *Better environmental conditions.* The noise levels in a hydraulic rock drill are noticeably lower than those generated by the pneumatic, owing to the absence of air escape. This mainly takes place in low frequency fields where protective earplugs are less efficient.

Apart from this, in underground operations the cloud of water and oil is not produced, improving the atmosphere and the visibility of the operator. On the other hand, hydraulic drive allows a more ergonomic design of the equipment, making the general work conditions and safety much more favorable.

– *More operative flexibility.* The pressure that drives the system in the drill can be varied, as well as the energy per blow and percussion frequency.

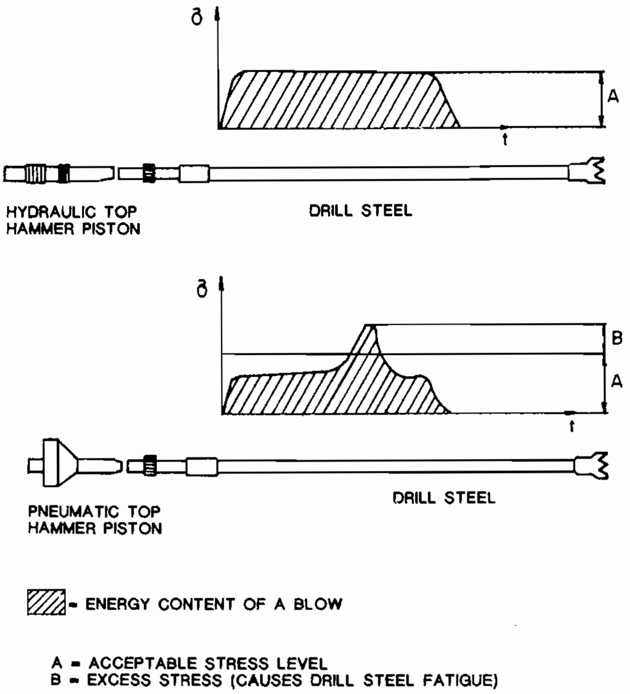

δ
A
t
HYDRAULIC TOP HAMMER PISTON
DRILL STEEL
δ
B
A
t
PNEUMATIC TOP HAMMER PISTON
DRILL STEEL
▨ - ENERGY CONTENT OF A BLOW
A - ACCEPTABLE STRESS LEVEL
B - EXCESS STRESS (CAUSES DRILL STEEL FATIGUE)

Fig. 2.20. Shape of shock wave and energy level in top hammer drilling.

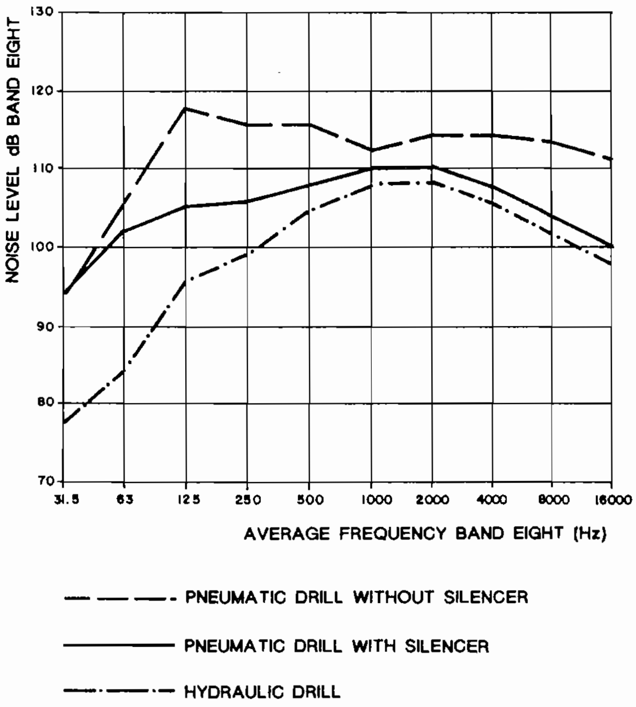


Fig. 2.21. Lower noise level produced by hydraulic rock drills.

– *Much easier to automatize.* This equipment is much more suitable for automatization of operations, such as changing drill steel, anti-jamming mechanisms, etc.

On the other hand, the problems that arise are:

– Higher initial investment.

– More complex and costly repairs than for pneumatic drills, requiring better organization and preparation of maintenance personnel.

2.4 DRILLING WITH DOWN THE HOLE HAMMER

These rock drills were developed in 1951 by Stenuick, and since then they have been widely used in surface operations with rocks of medium strength, and with diameters of 105 to 200 mm, although there are models which reach 915 mm.

The extension of this system to underground operations is quite recent, as it has become popular in this sector since 1975 on with the new Long Blasthole and Inverted Craters methods.

At the moment, in surface operations this drilling method is indicated for hard rocks and diameters above 150 mm, in competition with rotary drilling, owing to the strong development of hydraulic top hammer equipment.

The functioning of a down the hole hammer is based upon the fact that the piston delivers full impact directly on the drill bit. The driving fluid is compressed air that is supplied through a tube which serves as support and makes the hammer turn. The rotation is carried out by a simple pneumatic or hydraulic motor mounted on the surface rig, as also occurs with the feed motor, Fig. 2.22.

Flushing is carried out with the exhaust air of the hammer through the holes in the drill bit.

The blow frequency for down the hole hammers is usually between 600 and 1.600 blows per minute.

Today's down the hole hammer design is much simpler than the original one which had a butterfly valve incorporated to direct the air alternately to the top part of the piston. The valveless hammers are operated through ribbings or projections of the piston itself, allowing an increase in blow frequency, lowering air consumption and risk of dieselization.

To avoid entry of water, through effect of the hydraulic pressure, the hammers can have an anti-return valve installed in the air entry port.

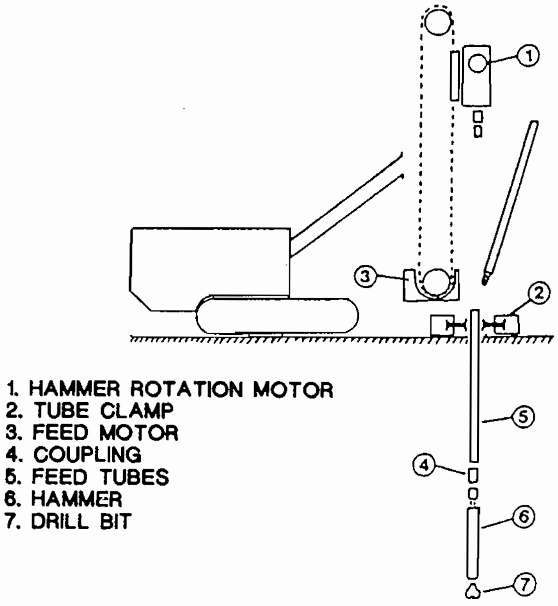


Fig. 2.22. Diagram of the components of a down the hole drilling rig.

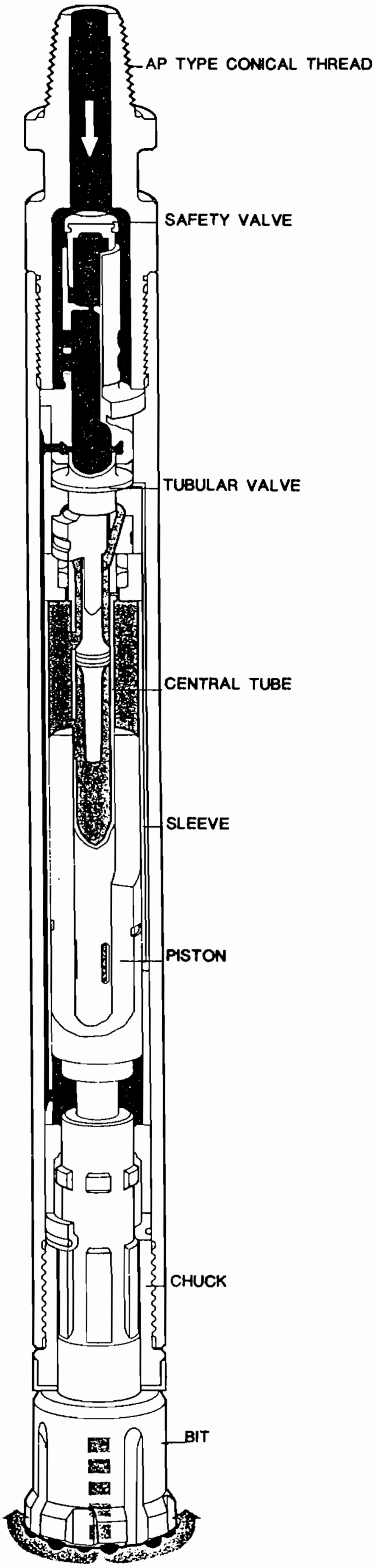


Fig. 2.23. A typical down the hole hammer (Atlas Copco).

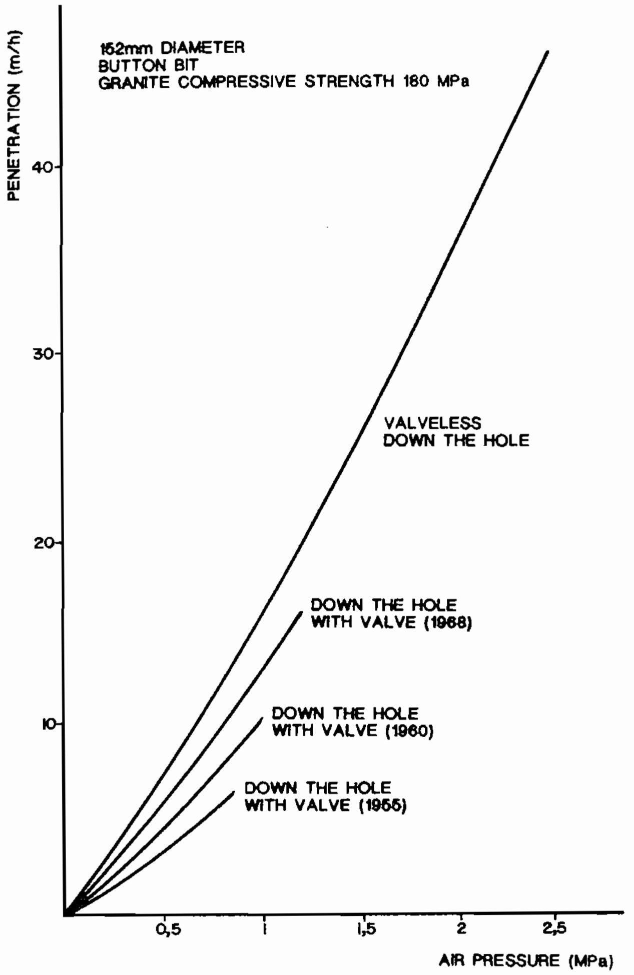


Fig. 2.24. Penetration rate and air pressures for different down the hole hammers.

The relationship stroke/diameter of the piston in top hammers is under or equal to 1, but in down the hole hammers, as the piston sizes are limited to the blasthole diameter, in order to obtain sufficient energy per blow, the relationship is around 1.6 to 2.5 for small diameters, and nearer to 1 for the large.

If the output power equation for a rotary-percussive drill is analyzed:

$$P_M = \frac{p_m^{1.5} \times A_p^{1.5} \times l_p^{0.5}}{m_p^{0.5}}$$

where: p_m = Air pressure acting on the piston, A_p = Effective area of the piston, l_p = Piston stroke, m_p = Piston mass.

It can be seen that air pressure is the parameter that has most influence on the penetration rate obtained with a down the hole hammer.

Nowadays there are valveless rigs that operate at high pressures, 2 to 2.5 MPa, with excellent yields.

In order to avoid void percussion, the hammers usually have a protection system that closes the air supply to the cylinder when the bit is not in contact with the rock in the bottom of the hole.

The drill bits are connected to the hammer by two systems: first, like a bayonet, the bit is inserted in the hammer and turned, usually to the left, which locks it in place: the second is by using retaining elements such as half-rings.

Table 2.6.

Type of rock	Rotation velocities (r/min)
Very soft	40-60
Soft	30-50
Medium	20-40
Hard	10-30

When a rock formation with the presence of water is drilled, a compressor should be available with sufficient air pressure to evacuate it when necessary. If not, the weight of the water column will reduce performance.

As to the pull down force that should be applied to maintain the drill bit in possible contact with the rock, a good rule of thumb is to come close to 85 kg per centimeter of diameter. Excessive thrust will not increase penetration; it will accelerate wear on the bit and put a heavier load on the rotation system. When drilling at high pressure, an additional feed force is needed in the beginning to overcome the effect of counter pressure from air in the bottom of the hole. The contrary happens when there is great depth and the number of tubes such that they go over the recommended weight. In this instance the operator must control the retention and rotation to maintain optimum thrust upon the drill bit.

The rotation velocities recommended in function with the type of rock are indicated in Table 2.6

A good rule is to adjust the rotation velocity to that of the advance with the following equation:

$$\text{Rotation velocity (r/min)} = 1.66 \cdot \text{Penetration rate (m/h)}$$

Apart from air, water can be used as flushing fluid, as well as injected foam. The latter has various advantages because it gives good results in large diameters with insufficient air, with low upward velocities (up to 0.77 m/s), and provides wall stability in soft formations. This method is especially indicated for water well drilling in unconsolidated ground.

Lubrication of down the hole hammers is of vital importance. Oil consumption varies with different models, but as a general rule 1 liter of oil per hour for every 17 m^3/min of air flow supply is recommended. When drilling at high pressure a minimum consumption of 1 liter/hour is suggested. If water or foam is used for flushing, the amount of oil should be increased.

As to tube dimensions, they should have a size that permits proper waste evacuation through the ring of space that is left between them and the blasthole wall. The recommended diameters as funtion of the drilling calibers are indicated in Table 2.7.

The advantages of down the hole drilling, in comparison to other systems, are:

– The penetration rate remains practically constant as the hole deepens, Fig. 2.25.

– There is less wear on the bit than in top hammer drilling, due to the flushing air that passes directly through the bit, cleaning the bottom of the blasthole efficiently and ascending through the small ring of space between the tube and blasthole wall.

Table 2.7.

Drilling diameter (mm)	Tubing diameter (mm)
102-115	76
127-140	102
152-165	114
200	152

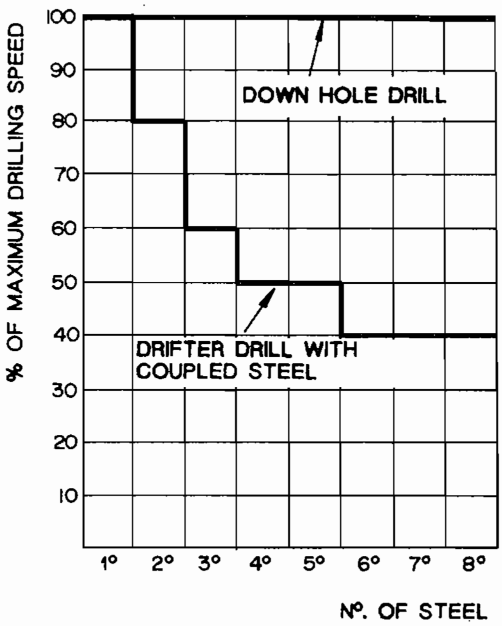


Fig. 2.25. Variation of the penetration rate with the number of steel in top and down the hole hammers.

– Longer tube life than that of the rods and sleeves.

– Very small blasthole deviation, making it ideal for longhole drilling.

– The lower energy per impact and the high blow frequency favors its use in decomposed formations or with unfavorable bedding.

– Less rotary torque needed than in other drilling methods.

– No loading bars are necessary and with small rigs it is possible to drill large diameter holes at great depths.

– The cost per lineal meter in large diameters and hard rocks is lower than for rotary drilling.

– Air consumption is lower than with top hammer pneumatic drills.

– The noise level in the work area is lower as the hammer is inside the holes.

On the other hand, the problems are:

– Low penetration rates.

– Each hammer is designed for a narrow range of diameters, that varies in 12 mm.

– The smallest diameter is limited by the hammer size with an acceptable yield, which at present is around 76 mm.

– There is a risk of losing the hammer inside the blasthole due to obstructions and detachment of the same.

– High pressure compressors are required with large energy consumption.

Table 2.8. Characteristics of some down the hole hammers.

Drilling diameter (mm)	100	125	150	200	300
Piston diameter (mm)	75	91	108	148	216
Piston stroke (mm)	100	102	102	100	100
Hammer weight (kg)	38.5	68.5	108	177	624
Air consumption (m^3/min cm)	4.7	6.7	10.1	17.1	28.2
Relationship hole diameter/Piston diameter	1.33	1.37	1.39	1.35	1.39
Relative air consumption (m^3/min cm)	0.47	0.54	0.67	0.86	0.94

At present, the down the hole system in the range of 76 to 125 mm is being replaced by hydraulic top hammer drilling.

In Table 2.8, the technical characteristics for some down the hole hammers can be found.

2.5 ADVANCE SYSTEMS

As indicated before, to obtain high performance from the drills, the bits must be in contact with the rock and in proper position at the moment that the piston transmits its energy by impact. To achieve this, in manual drilling as well as in mechanized, a force should be exerted on the drill bit that oscillates between 3 and 5 kN, for small equipment, and more than 15 kN for large drills.

The advance systems can be the following:

- Airlegs;
- Chainfeed;
- Screwfeed;
- Cablefeed;
- Hydraulic feed.

Telescopic airlegs are used in horizontal blasthole drilling as well as for vertical, where they are called stoper leg, or column airlegs.

2.5.1 *Airlegs*

Basically, an airleg consists of two tubes, an outer tube of aluminium or another light metal, and a inner tube of steel that is connected to the jackhammer. The inner tube acts as a piston with double effect, controlling its position and thrust force with a valve that is connected to the compressed air circuit, Fig. 2.26.

2.5.2 *Chainfeed*

This system consists of a heavy gauge chain that runs between two channels and is powered by a pneumatic or hydraulic piston type motor, depending upon the fluid used to drive the hammer, by means of a worm and a wheel type gearbox.

The chain is attached to a travelling drill mount which runs over the top of the channels.

The advantages of this system, which is widely used in surface as well as in underground equipment, are: its low cost, easy repair and the possibility of long feed lengths. On the contrary, there are some problems, such as: more wear in abrasive conditions, danger if the chain breaks

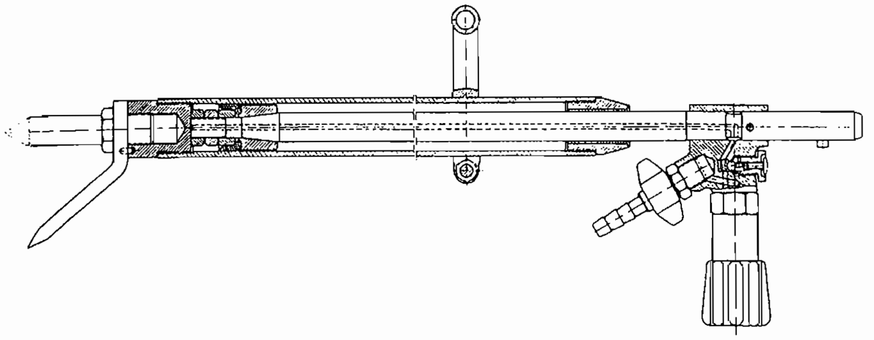


Fig. 2.26. Cross section of a pneumatic airleg (Padley & Venables).

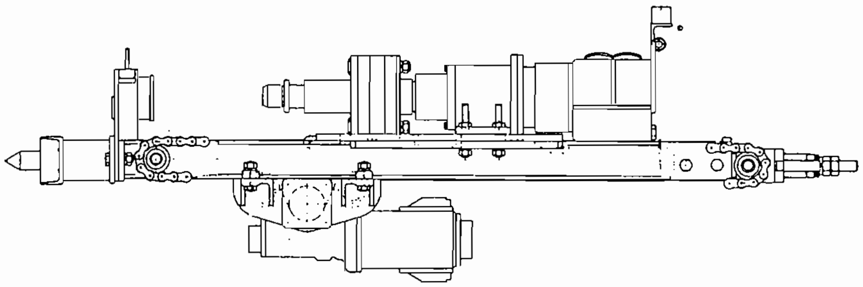


Fig. 2.27. Chainfeed (Padley & Venables).

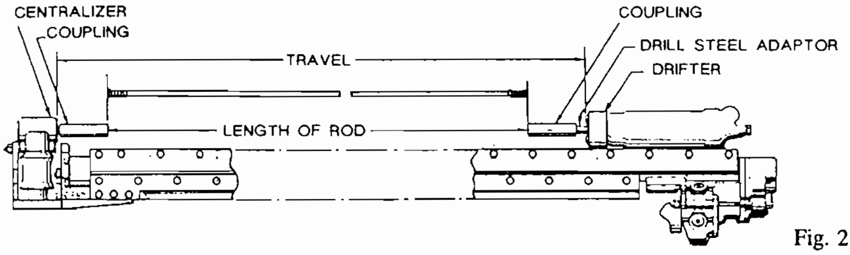


Fig. 2.28. Hydraulic drifter feed.

while drilling upwards, and difficulty in obtaining a smooth feed when the penetration rate is low.

2.5.3 *Screwfeed*

In this system, the feed thrust is produced by the rotation of a lead screw run by a piston or gear type air motor. The screw has a small diameter in relation with its length and is subject to bending forces and vibrations during drilling. For this reason, lengths over 1.8 m are not common.

The principal advantages of this method are: a regular and smooth feed force, great resistance to wear, very powerful and adequate for deep blastholes, less bulky than chainfeeds and safer.

However, this system also has inconveniencies which are: high price, difficult to repair and limited in length.

2.5.4 *Cablefeeds*

In Canada this system is very popular and is mounted on special jumbos called Stopewagons. Basically, it consists of a hollow light extrusion upon which the drilling machine slides. A piston moves inside and is connected at each end to a cable that comes out of each end of the extrusion through closures or glands. The piston is run pneumatically.

Its advantages are: low price, easy repair, robust and long operational life.

The disadvantages are: limited to small machines and short blastholes, the air losses through the glands at the end of the extrusion and the danger if the cable breaks.

2.5.5 *Hydraulic feeds*

The rapid development of hydraulics in the last decade has widened the use of this type of feed to pneumatic drills.

The system consists of a hydraulic cylinder that moves the drill along the support boom. Fig. 2.28.

Hydraulic feeds give the following advantages: simple and robust, precision and easy to control, very powerful and capable of drilling deep blastholes and its adaptability to a large variety of machines and blasthole lengths.

To the contrary, the disadvantages are: higher prices, requires a separate hydraulic supply, less adaptable to percussive drills than to rotary, and wear on the cylinder that pushes.

2.6 MOUNTING SYSTEMS

In the types of work carried out by drilling equipment, the mounting systems can differ depending upon the application, either underground or surface.

In this section, the handheld drills will also be discussed.

2.6.1 *Mounting systems for underground applications*

The most commonly used drilling rigs for underground labors are the following:

– Jumbos for driving of tunnels and drifts, cut and fill operations, chambers and pillars, etc.

– Drills for longholes in fan cut for sublevel and block-caving operations.

– Longhole drills for vertical crater retreat systems and sub-level open stoping.

Other units such as the rubber tire wagon drills, and crawler drills are described in detail along with the surface machines.

Special mountings for sinking shafts or drilling raises are discussed in another chapter.

A. *Jumbos*

Jumbos are drilling units equipped with one or various drill hammers and their principle application in underground labors are:

- Driving of tunnels and drifts.
- Bolt hole drilling and cross cutting.
- Benching with horizontal blastholes.
- Cut and fill mining.

The basic components of these rigs are: the translation system, the drive system, booms, screw or chain feed and hammers, Fig. 2.30.

These machines can be towed or, more commonly, self-propelled. The latter are rubber tire, crawler or rail mounted.

The first is the most used for its great mobility (up to 15 km/h), for its resistance to corrosive waters and for the least wear over irregular ground. The chasis are articulated in most instances, permitting excavation operations with curves.

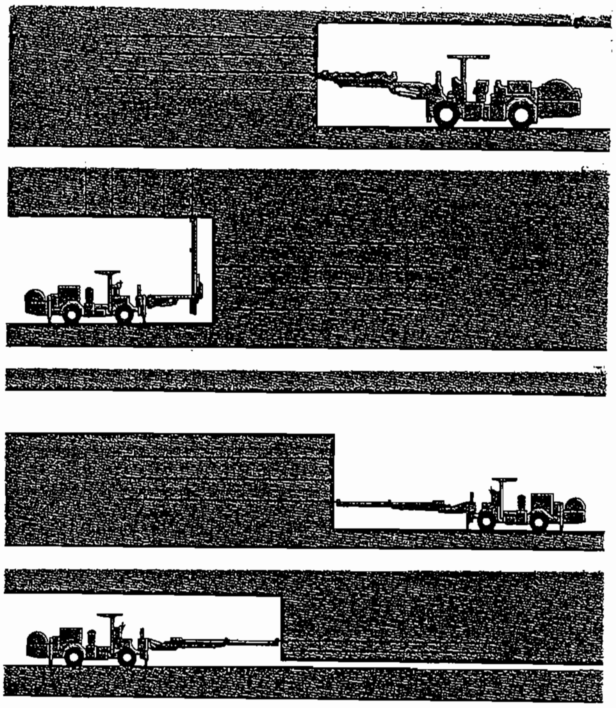


Fig. 2.29. Jumbo applications.

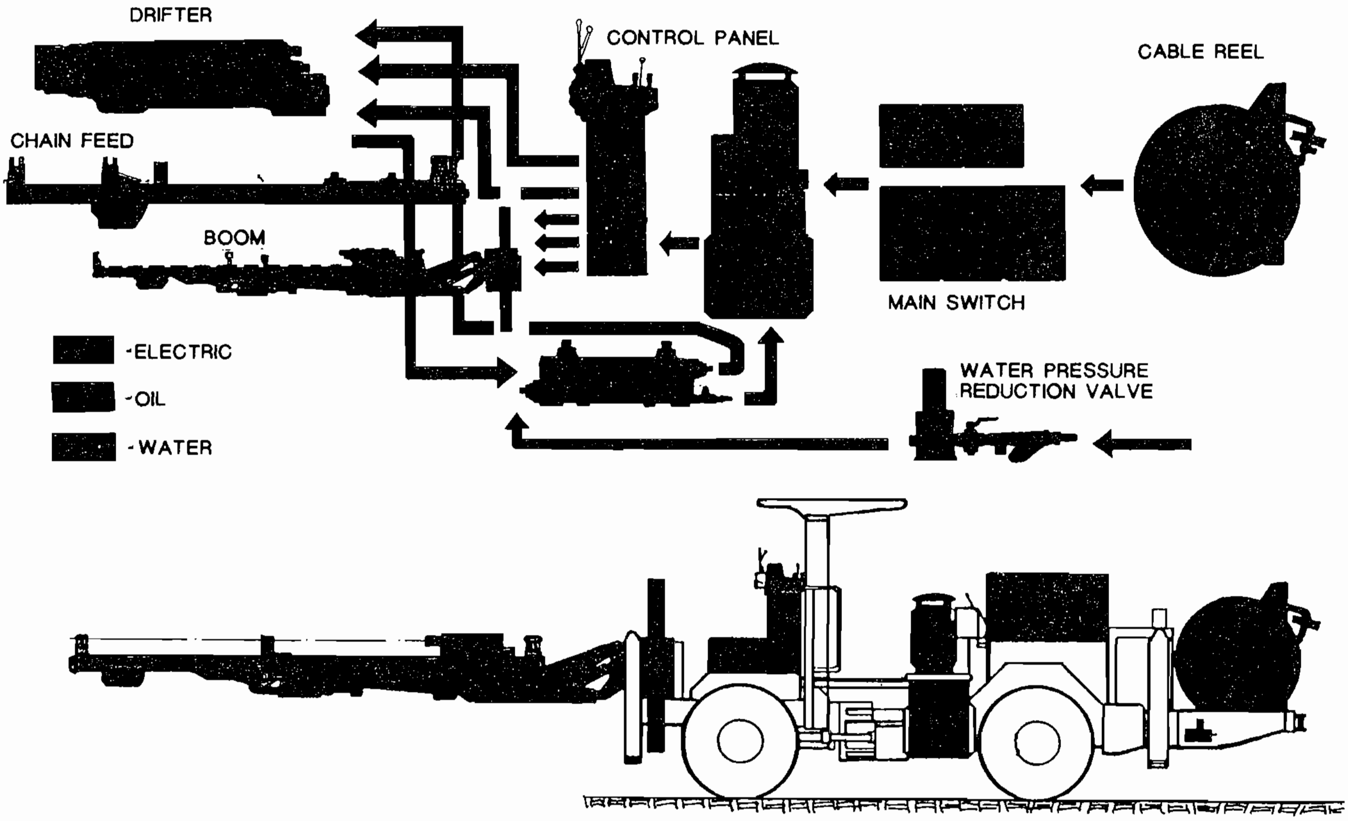


Fig. 2.30. Principal components of a Jumbo (Tamrock).

The crawler mounted jumbos are used over very bad terrain, narrow drifts, with high slopes (15 to 20°) and few curves, Fig. 2.31.

The jumbos mounted on rails, which are rarely used nowadays, find application when the work is: of great length, small section, poor ventilation and the loading and haulage equipment is also mounted on rails. With these rigs it is imperative that from each position the jumbo can drill all of the forseen blastholes, Fig. 2.32.

The sources of energy can be: diesel oil, electric or compressed air. The diesel engines that drive the rolling mechanism, by mechanical or hydraulic transmission, can also be used to operate all the drilling elements, including the compressor and hydraulic units. This system is used in small projects and where there are no contamination problems. It is more often found that the diesel motor is used to move the rig from place to place and an electric motor to operate the drilling equipment. In this case an electrical installation is necessary to distribute the energy. Compressed air is only used when there is a network in good condition, otherwise this system is rejected.

The booms of modern jumbos are hydraulically operated and there is great variety in design, but they can be classified in the following groups: tripod type, base rotation or roll-over boom. The coverage and work possibilities of the jumbos depend upon the number of cylinders and movement of the boom, for which the choice of booms is a very important aspect, above all in civil engineering, more than in mining, as the work is more varied.

There are also booms with telescopic extension that can be extended from 1.2 to 1.6 m.

The number and size of the booms goes in hand with the required advance, tunnel section, and drilling control to avoid subdrilling.

The following should be observed as general criteria: the number of blastholes to be drilled should be approximately the same for each boom, the overlapping area of each boom should not be more than 30%, and the blasthole drilling order should be such as to permit the least moving time possible for the booms.

In order to calculate the number of booms per jumbo and operator and their work output, the following formulas can be used:

$$N_b = \frac{L_v \times e}{VP \times t_m}$$

$$P_j = \frac{60 \times L_v \times N_b \times e}{\dfrac{L_v \times t_b}{l_b} + t_m + \dfrac{L_v}{VP}}$$

where: N_b = Number of booms per operator, P_j = Production capacity per operator (m/h), L_v = Length of single pass feed (m), VP = Penetration rate (m/h), t_m = Retract, repositioning of the rod and collaring time (1 – 2 min), t_b = Bit change time (1.5 – 3 min), l_b = Meters of blasthole per bit (m), e = Operator's efficiency (0.5 – 0.8).

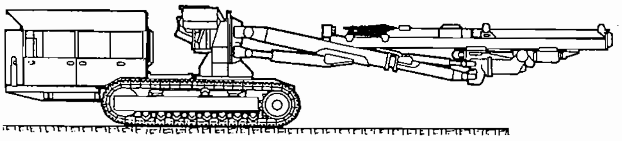


Fig. 2.31. Crawler mounted jumbo (Atlas Copco).

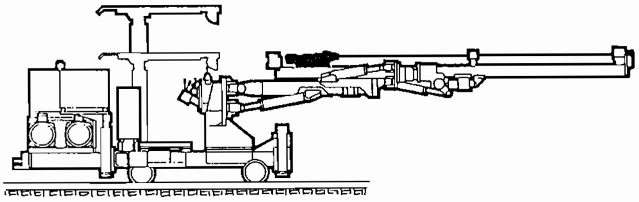


Fig. 2.32. Rail-mounted Jumbo (Atlas Copco).

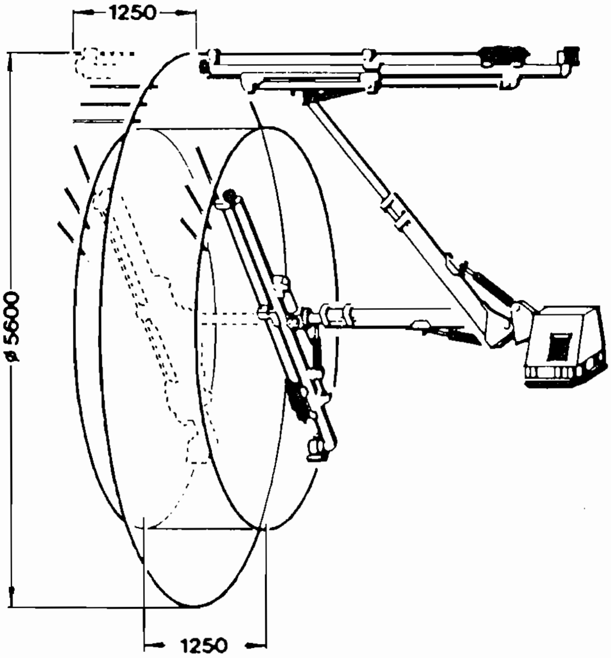


Fig. 2.33. Base rotation boom (Atlas Copco).

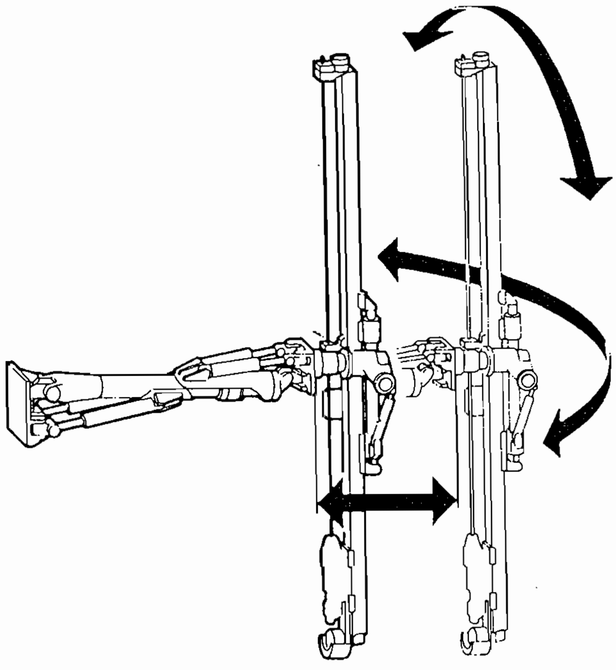


Fig. 2.34. Extendable booms with roll-over head.

The feeds can be of the type described before, preferably chain or screw feed. They are lighter than those used for surface operations and have an advance motor in back to avoid blows. Apart from the end centralizers, there are also intermediate centralizers to eliminate bending of the drill steel, as it is usually very long and thin.

As it is not normal to add rods for the drilling of a round, these can reach 4.20 m in length or even longer. When the operator has to control several blastholes, the feed control can be automatic with detention of drilling at a prefixed depth or the hammer reaches the end of its stroke on the feed. It is also normal to install an automatic system of parallelism to eliminate deviations through angulation errors, and a medium power collaring device.

The drills are usually rotary-percussive or rotary, depending upon the type of rock to be blasted, the drilling diameter and the required output. These drills differ from those of surface operations in that they have a lower profile to be able to correctly drill the contour blastholes without an excessive inclination which would produce saw tooth irregularities. For this reason, the rotation systems of the hammers usually go in the opposite position of those on the surface, remaining inside the feeds.

The drilling diameters depend upon the section of the tunnels or drifts, and for a rock of medium to hard strength they can be established as indicated in Table 2.9.

As for these diameters the drill steel, whether of fixed length or extendable, has diameters between 25 and 37 mm, the underground drills are much lighter than those for surface operations with lower energies per blow and higher impact frequencies.

The hammers tend to be hydraulically operated, substituting the pneumatic, for all the advantages described before, to which must be added a lower contamination by oil mist and the elimination of ice problems in the exhausts.

Portal jumbos are used for the drilling of large tunnels or chambers, Fig. 2.35. These structures are designed for a specific job and have an opening, or portal, through which the loading and haulage equipment can pass to remove the blasted material. The advantage is that the jumbo does not have to be pulled out of the way, only moved to a safe distance when blasting.

These jumbos can have several booms, as well as the man baskets that are operated hydraulically to give easy access to the blasters when charging the holes or during support operations.

B. *Longhole fan drilling*

In underground metal mining the method known as sublevel caving is frequently used. For breakage with explosives, precision blastholes of 20 to 30 m must be drilled, in fan pattern on a vertical or inclined plane, and can be done either upward or downward.

Table 2.9.

Section to be excavated (m^2)	Drilling diameter (mm)
< 10	27-40
10-30	35-45
> 30	38-51

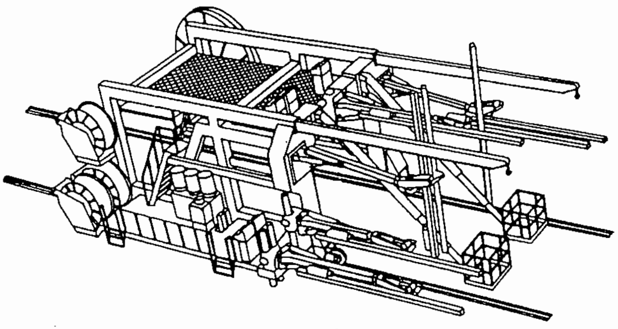


Fig. 2.35. Portal type jumbo (Tamrock).

In the beginning, pneumatic hammers were used with diameters between 50 and 65 mm. The drilling outputs and breakage productivity were quite low.

The equipment still in use have hammers mounted upon feeds, usually screwfeed, that are held by a balance beam or crowns anchored to a transveral bar that gives coverage to a whole fan drilling pattern from one position.

The smallest rigs are installed on a roller or skid that is connected to a control pannel, and the medium sized are mounted on self-propelled pneumatic wagons.

The units have remote control for the drills, as well as the line greasers and support devices on the excavation walls to avoid unnecessary movement of the rig.

Lately, the use of hydraulic hammers and heavy drill steel has permitted the use of 102 to 115 mm diameters, making these work methods interesting again, as they had lost ground when compared to other alternatives such as vertical crater retreat or sub-level open stoping.

The largest equipment has an electrohydraulic drilling system similar to that of the jumbos on rubber tyres, and a diesel motor for translation, which can even operate the hydraulic power unit.

The frames are usually rigid and mounted on crawlers or rubber tyres, although articulated units on tyres also exist. The feeds vary according to the manufacturer and can be chain, screw or telescopic cylinder. These feeds can be moved laterally to drill parallel blastholes, or turn through 360° for fan drilling patterns.

In order to achieve a safe and firm positioning during collaring and drilling, there are pneumatic cylinders that push against roof and floor to anchor the rig.

C. *Drilling large diameter longholes*

The use of the Vertical Crater Retreat (VCR) method and its derivative of Long Blastholes, created a revolution in metal mining some years ago, as it allows the drilling of large diameters and patterns, which means high output and productivity, as well as low breakage costs.

Drilling is carried out with diameters of 100 to 200 mm and usually with high pressure down the hole hammers with which interesting penetration rates are obtained.

Although some machines are mounted on rubber tyres, the frames with crawlers are used more frequently. The main difference between these rigs and those used for

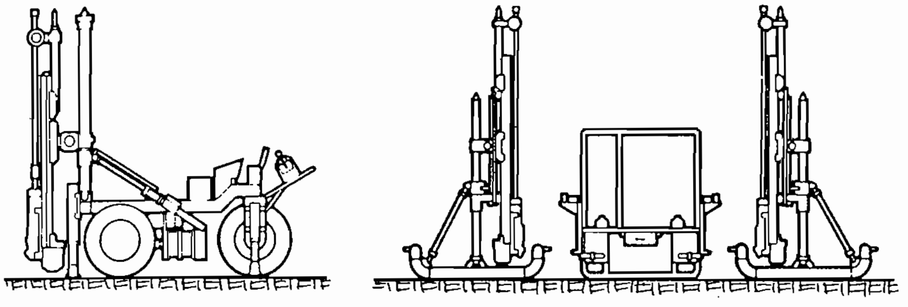


Fig. 2.36. Wagon drills and equipment on skids for drilling long blastholes.

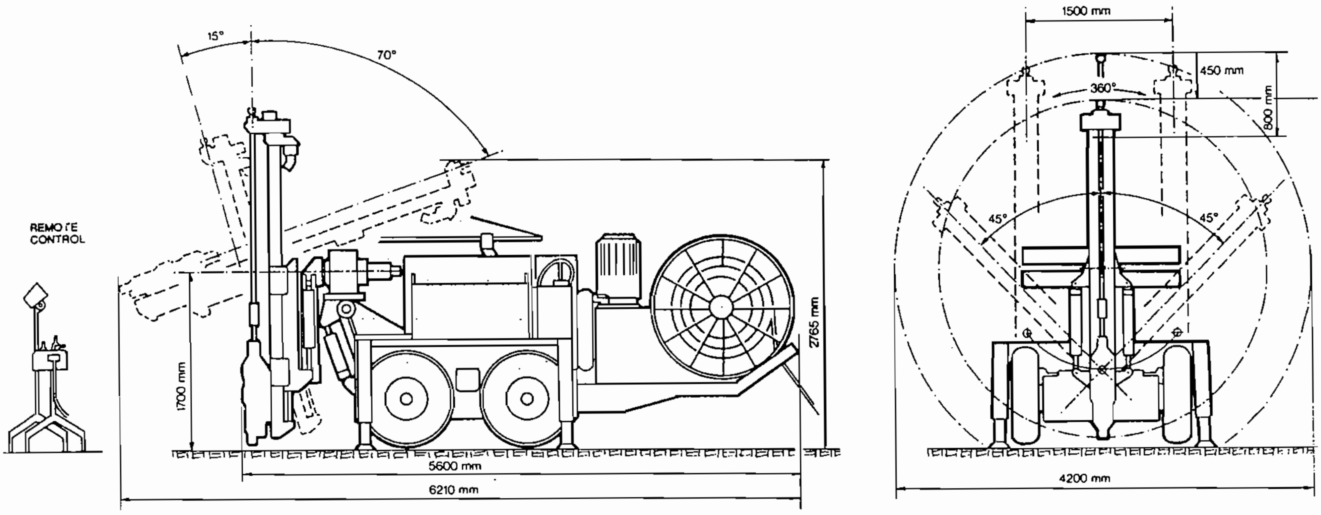


Fig. 2.37. Drilling rig for long blastholes mounted upon a frame with rubber tyres (Atlas Copco).

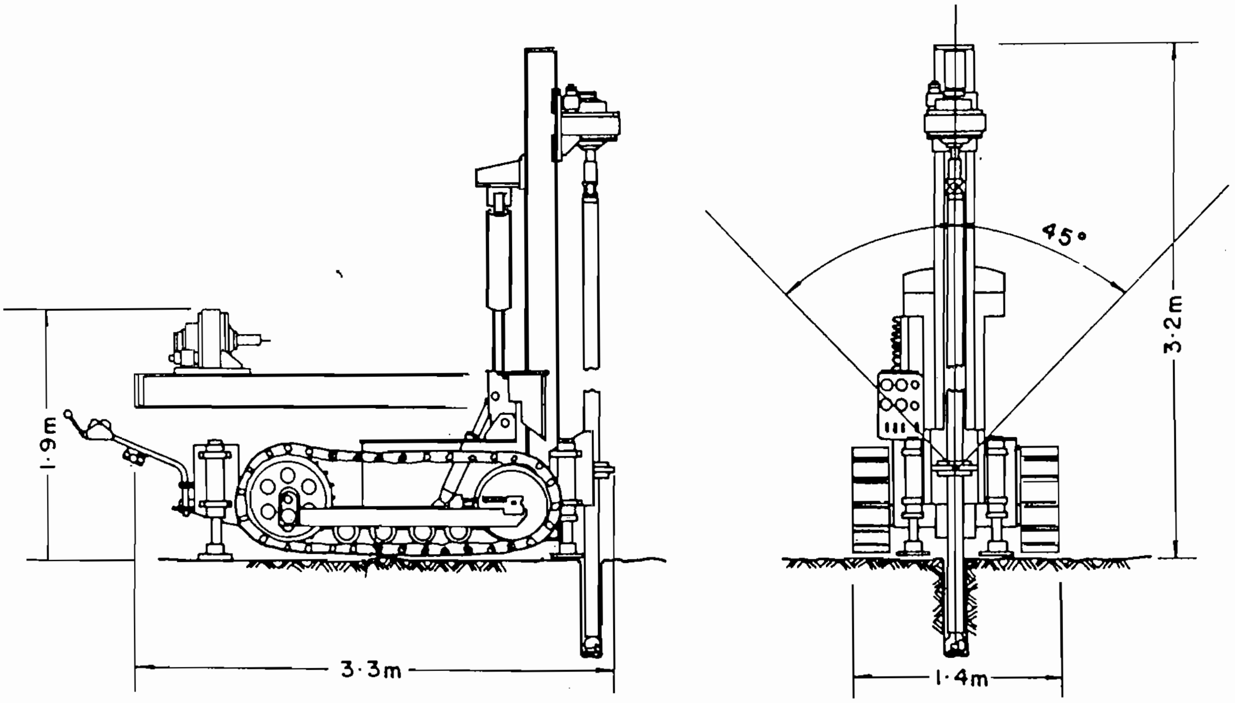


Fig. 2.38. Underground drilling rigs with down the hole hammer (Ingersoll Rand).

surface operations are:

– More compact design with a shorter and more robust feed, and the advance system is of hydraulic cylinder or chainfeed.

– They have hydraulic leveling jacks.

– The rotation head gives excellent rotary torque and ample control over the rotation velocity.

Apart from production blasthole drilling, they are used in other jobs such as: drainage openings, ventilation, hydraulic fillings, electric lines, shift and tunnel driving, as well as the advance for raises.

2.6.2 *Carrier systems for surface applications*

In surface operations, the carrier systems for drills are: light chasis on tyres, crawler wagons and on trucks, Fig. 2.39.

The first attempts to mechanize work in quarries consisted in using wagon drills on wheels. These wagons have a small u-shaped frame with two fixed wheels and a third that revolves with a steering bar which is used as the tow bar for transport. The drills are mounted on feeds which are attached by clamps to a round bar or transversal support positioned at the end of the main boom. They can be positioned at any angle by manual adjustment of the clamps and can turn on a vertical plane, Fig. 2.40.

When down the hole hammers are used, the pneumatic rotation motor is placed upon the feed. The main inconvenience of these rigs is the time lost in positioning and translation.

The most popular carrier system is that of crawler drills. At present time there are two groups of rigs: pneumatic and hydraulic.

The pneumatic rigs, which are the oldest, have the following principal components:

- Crawler mechanism,
- Translation motors,
- Chasis,
- Auxiliary hydraulic power unit,
- Advance engine, and
- Hammer.

The crawlers are independent and carry a hydraulic cylinder in each one which are interconnected to absorb the oscillating movement, avoid blows when creeping over rough ground and allow drilling from difficult positions, Fig. 2.42. The height from the ground is an important design criterium to overcome obstacles during creeping when changing positions.

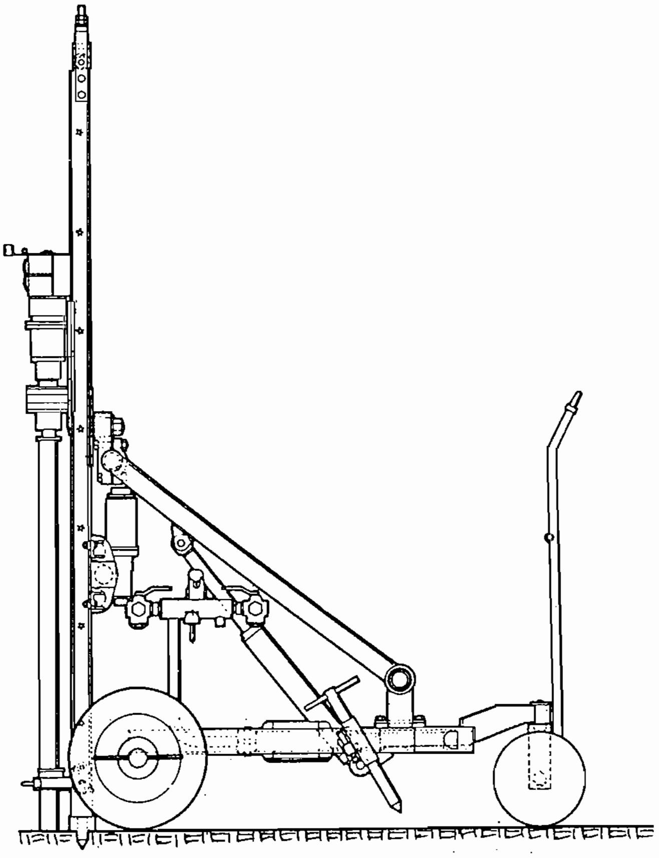


Fig. 2.40. Drill wagon on rubber tyred wheels.

The traction engines are independent and operated pneumatically, of piston type, with covered gears connected to controls and disc brakes.

The hydraulic cylinders of the booms and the feeds are operated by a hydraulic pump run by a pneumatic motor.

The booms of these rigs can be of fixed length or extensible and articulated, and they are anchored to a place on the frame. In Fig. 2.43, the principal movements that can be carried out by the booms and feeds of these units are shown.

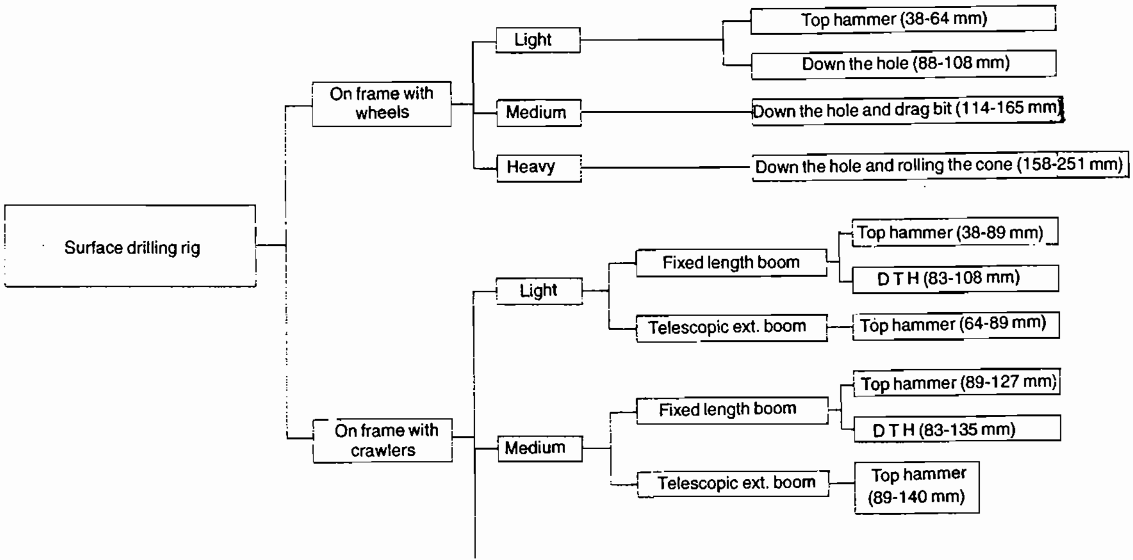


Fig. 2.39. Carrier systems for surface drills.

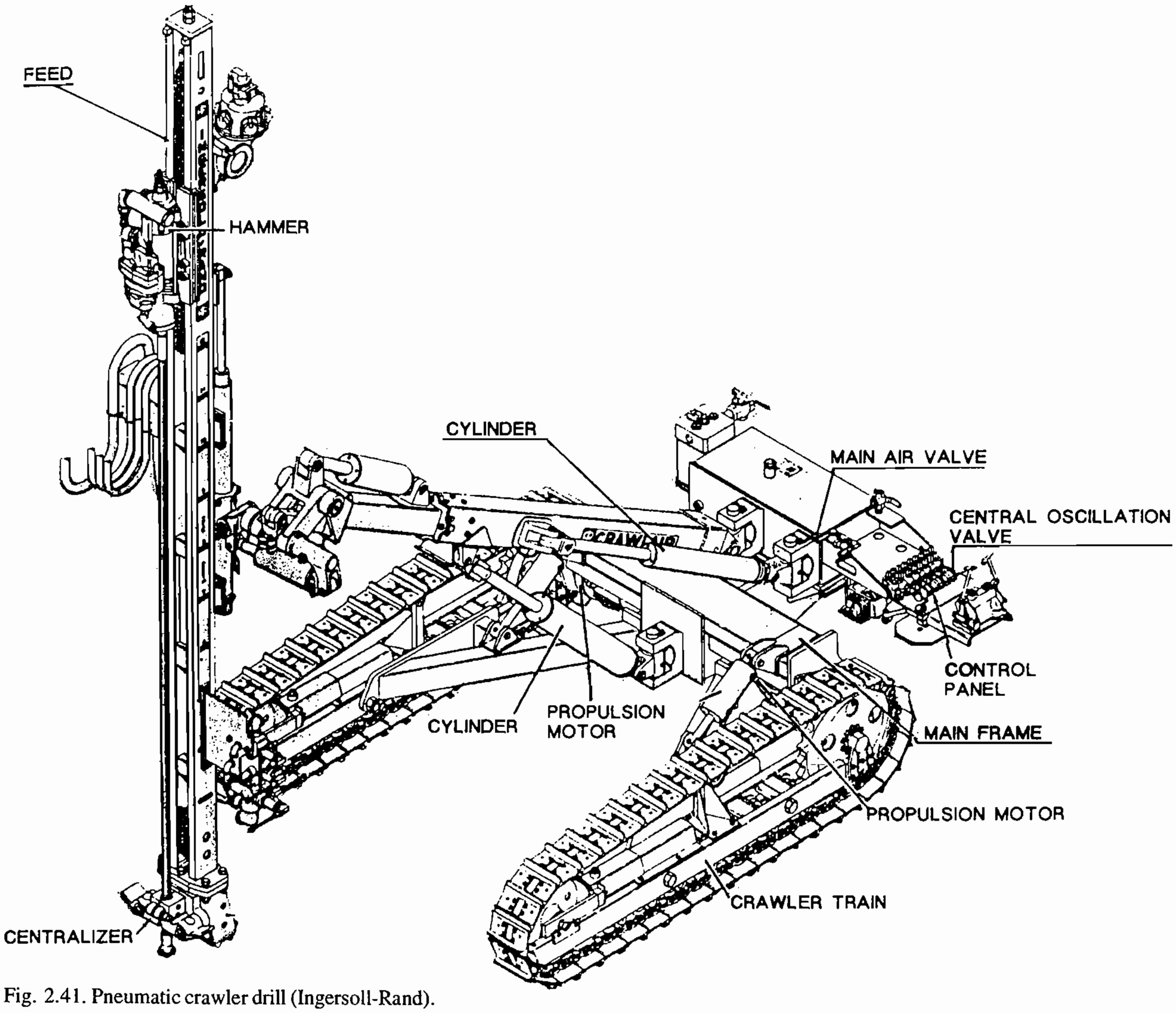


Fig. 2.41. Pneumatic crawler drill (Ingersoll-Rand).

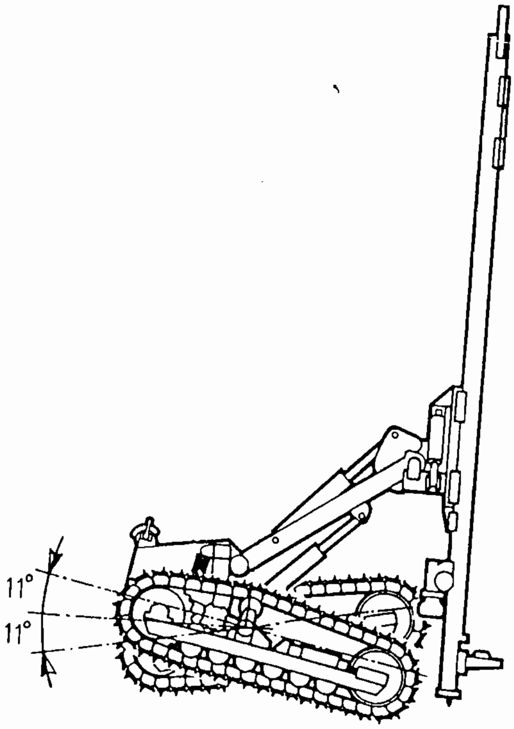


Fig. 2.42. Oscillating movement of crawlers (Atlas Copco).

Photo 2.1. Pneumatic traction engine (Ingersoll-Rand).

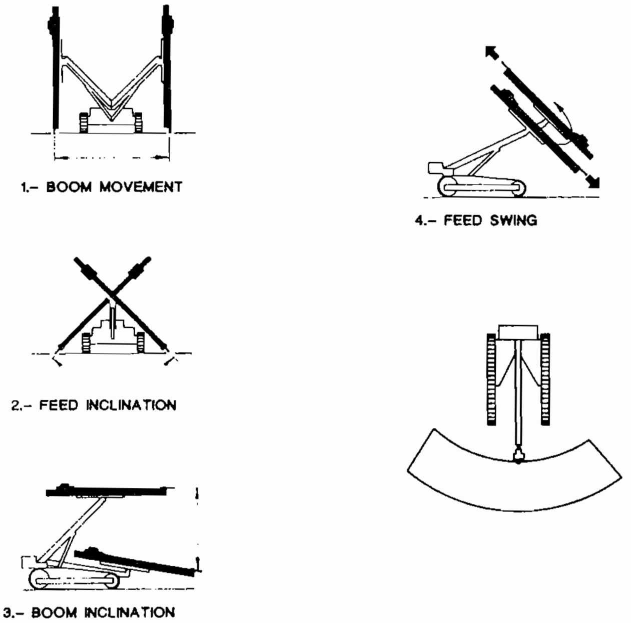


Fig. 2.43. Movements of the boom and feed.

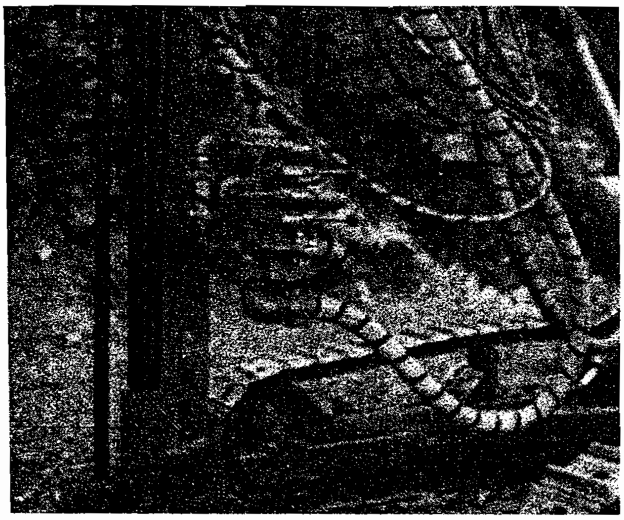


Photo 2.2. Control pannel for the drill, centralizer and drill steel support (Ingersoll-Rand).

The most sophisticated booms are usually found in civil engineering operations, as in mining the jobs are more rutinary and over more uniform surfaces.

The feeds have: an advance motor, a hammer or roll-over head, controls for drilling, centralizer and support for the drill steel.

When drilling with top hammers these are mounted on the feeds; and with down the hole hammers the rotation heads are mounted on the feeds.

The centralizer or clamp-guide assures the proper beginning of the blastholes and makes the changing of drill steel possible.

The control pannel is usually installed on the feed and permits selection of the most adequate drilling parameters for each type of rock.

The controls for the traction engines and towing winch are usually placed on a rotating boom that allows the operator to stand away from the machine and move it under safer conditions.

These rigs have a hook on the back end to pull the compressor.

When down the hole rock drills are used, in order to reduce air consumption, they are hydraulically driven in the following components: translation motors, advance motors, rotation heads and movements of boom and feed. The energy savings is high, as indicated in Fig. 2.44.

The drilling rigs that are totally hydraulic have the following advantages:

- Lower powered installation, therefore lower fuel consumption.
- Robust and compact design which usually carries a flushing compressor on board.
- High transfer speed and excellent maneuverability.
- Wide range of drilling diameters, from 65 to 125 mm, with equipment on the market that work between 200 and 278 mm.

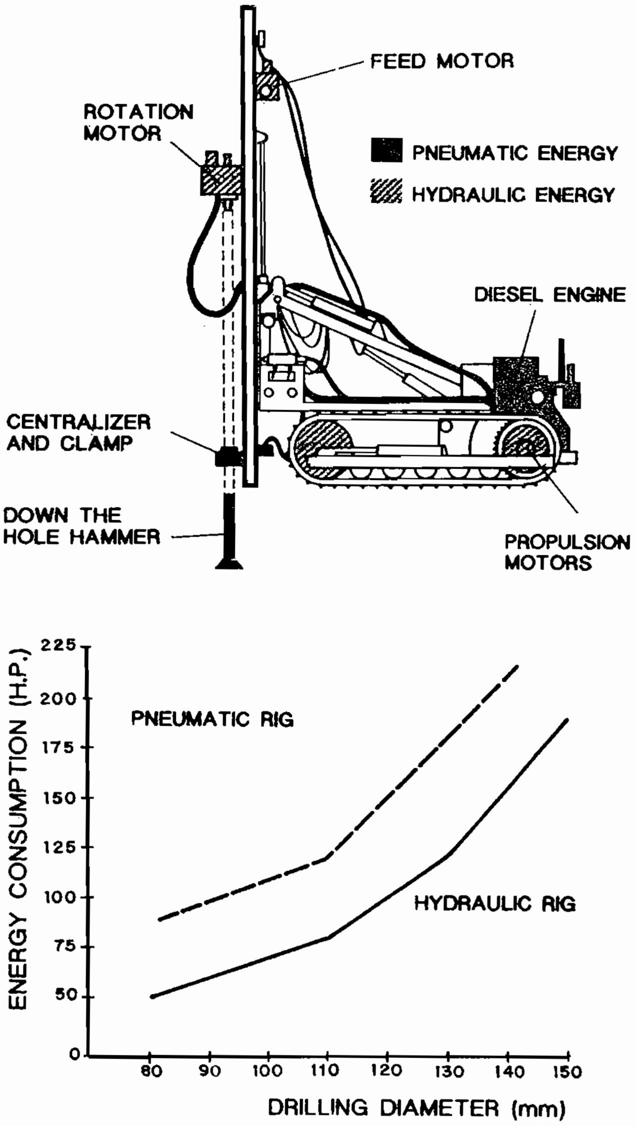


Fig. 2.44. Energy savings in down the hole pneumatic drills with hydraulic drive.

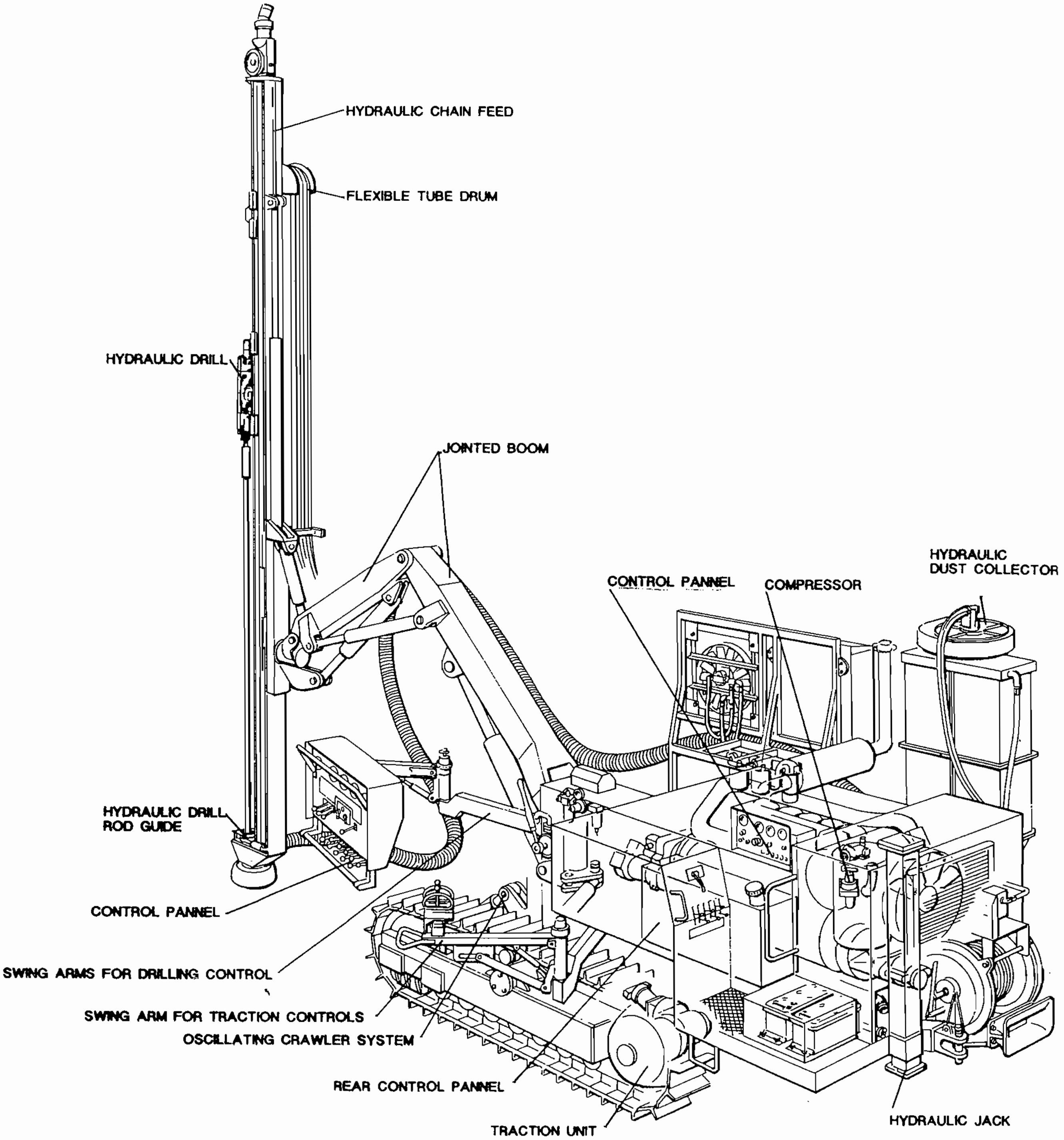


Fig. 2.45. Hydraulic rig (Atlas Copco).

- Possibility of incorporating an automatic drill steel rod changer.
- Drilling rates that are from 50 to 100% higher than with pneumatic rigs.
- Better environmental conditions.
- Lower drilling costs.

On the other hand, the following problems exist:

- Higher capital cost.
- A more qualified and careful maintenance is required.
- More difficult to repair than the pneumatic rigs.

As far as design goes, they are quite similar to the pneumatic rigs; however there are a few differences such as:

- The source of energy is usually a diesel engine, although electric units do exist that run the hydraulic power unit and the flushing air compressor.
- The hydraulic pumps, normally four, are of fixed flow, although there are pumps with variable flow on the market.
- The maximum hydraulic fluid pressure is usually under 20 MPa.
- They have the following optional elements: dust collector, sound-proof and climatized operator cabins,

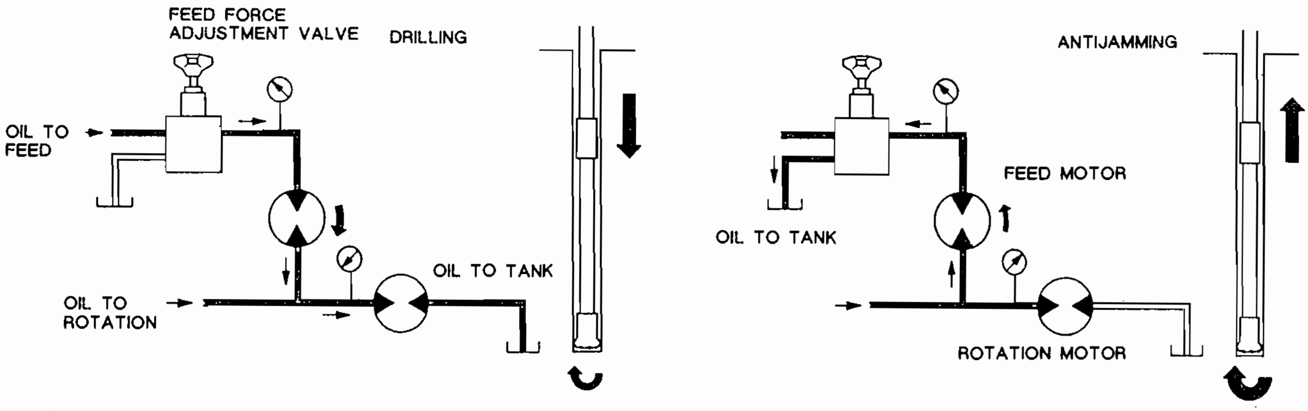


Fig. 2.46. Anti-jamming automatics (Tamrock).

automatic drill steel changers and hydraulic jacks.

– Most of the manufacturers incorporate an anti-jamming system, Fig. 2.46.

– The crawlers have hydraulically adjustable tensors.

– The traction motors usually are of inclined axial piston type with fixed, symmetrical displacement in order to be able to turnover in both directions.

– The feeds have a movable drum for rolling up and directing the hydraulic hoses.

– The hydraulic advance motors have maximum feed/pull out force between 20 and 32 kN, with advance speeds of up to 40 m/min.

– The drill steel guide is hydraulic, as well as its stop.

– The fuel deposit has sufficient capacity for operation during one, or two shifts, in some cases.

Using a truck as carrier is only done with rotary rigs and/or down the hole hammer that has high pressure compressors.

On occasions, small, multi-purpose front end loaders are used, with a retro-boom upon which a drill is mounted.

These units are able to drill blastholes of 22 to 89 mm in diameter, with integral or extensible drill steel. The jobs that are most frequently performed are: secondary drilling, ditches, foundations, etc.

2.6.3 *Handheld drills*

Underground and surface handheld drills are, in concept and method of work, similar, and only differ in small details.

The handle of the surface drills is open, to hold the hammer with both hands, whereas the underground model is closed and for only one hand in order to adapt the drill to horizontal holes.

For the first, the operating and flushing is totally pneumatic, not so in those for underground where the flushing can be carried out with air and/or water. The water pressure should always be lower than that of the air to avoid flooding the hammer.

Handheld drills are normally fitted with a latch-type retainer for the drill steel. They are of integral type with drilling diameters of 22 to 45 mm and lengths of 400 to 6400 mm.

The designs differ in the valve systems utilized, either rocker or tubular, and the rotation mechanism, whether fluted rifle bar or ratchet wheel. In function with weight, they can be classified as light, medium or heavy (20, 30, or 40 kg). The air consumption oscillates between 50 and 100 l/s and the piston sizes and their length of stroke varies between 65 to 80 mm and 45 to 70 mm, with a blow frequency of 30 to 50 per second.

Mufflers, or silencers, can be placed around the casings of the cylinders to absorb the exhaust noise; they hardly affect the drilling rates and they reduce the noise level by some 7dB.

The most important applications in surface operations are: pop and toe shooting, drilling in small size operations, demolitions, etc. In underground projects, apart from secondary drilling, they are used for production drilling and also in small cross section tunnels and drifts of short length, where an investment in mechanized equipment is not justified. In these cases it is normal to work with airlegs for horizontal drilling and with advance cylinders when the holes are vertical.

2.7 DUST COLLECTORS

The elimination of dust during drilling is carried out for two reasons; one, to improve work conditions, and two, to increase productivity.

Drilling dust, especially if the rock has a high silica content and its size is under 0.005 mm, constitutes a risk for operators, thus many countries have safety and health regulations that make its elimination obligatory.

Other technical and economical reasons that justify the use of dust collectors are:

– Lower maintenance of the motor-compressor equipment, giving higher mechanical output.

– Higher drilling rate (about 2 to 10% more), due to the fact that the cuttings are removed from the blasthole, therefore avoiding their regrinding. Besides, the operator can be closer to the controls of the machine, increasing the efficiency and control of drilling.

– The drilling costs are lower, because of higher output as well as less wear on the bits.

– There is the possibility of taking samples of the

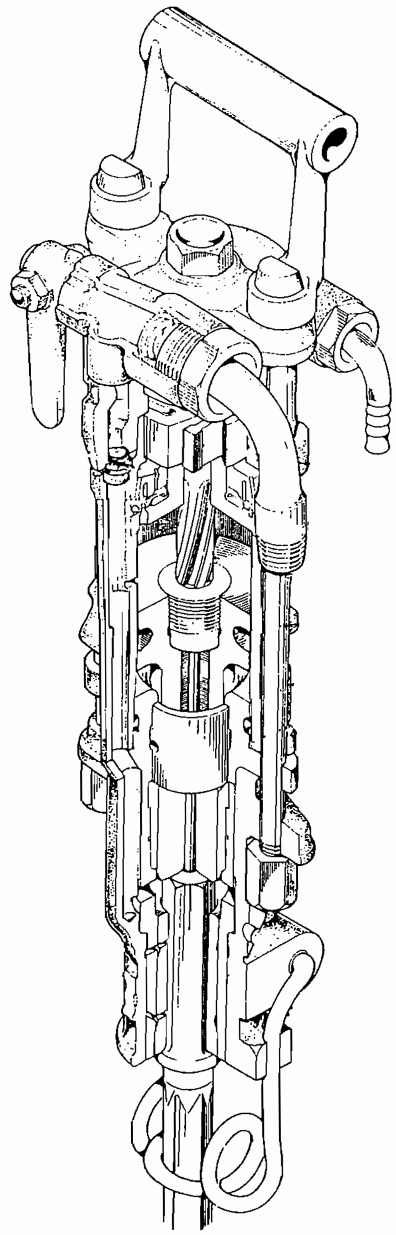


Fig. 2.47. Handheld drills (Gardner-Denver).

drilled rocks in order to control the laws and planning.

At present time, all drilling equipment can be used with dust collectors, including handheld drills (jackhammers). This has noticeable advantages over the systems of water injection, or water with foam, which are only justified when water is present in the rock formations.

Dust collectors basically consist of:

– A suction head that is placed on the surface at the blasthole collar where the dust is aspirated and sent through the suction hose to the separation and filter unit.

– A system of separation and filtering. It is carried out in two stages: first with cyclone, where bigger particles and most of the dust is separated by means of centrifugal force, and secondly, the filtering is carried out that retains the dust particles under 5 μm.

– A depression or partial vacuum system, with a fan placed behind the filter unit that is operated by hydraulic or electrical energy, or pneumatically on occasions.

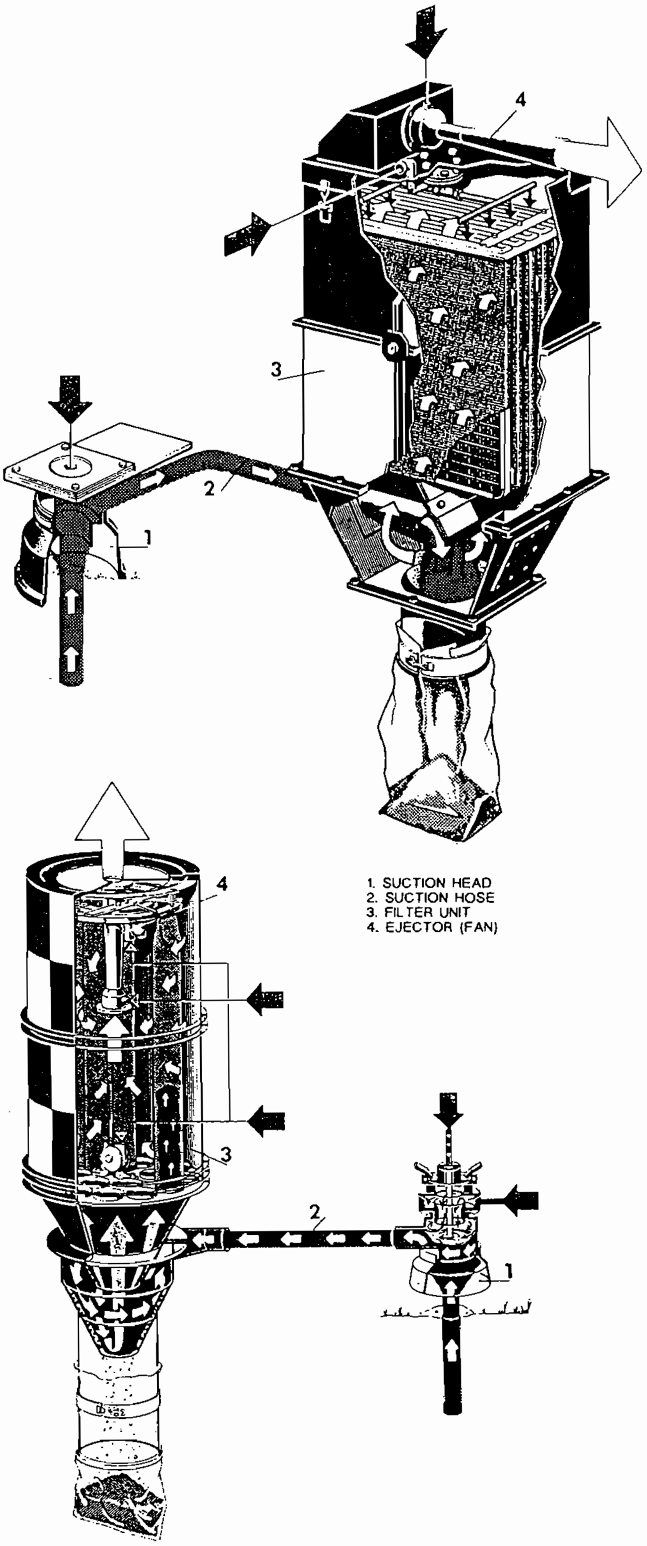


Fig. 2.48. Dust collection system (Atlas Copco).

The suction head has two openings: one on top to allow passage of the drill steel, and another larger one on the bottom where the flushing air passes with the cuttings and dust. The design of the suction head collector should avoid the escaping of any air when the drill dust expands. This is achieved in small equipment by the suction of the fan, and in large equipment by an ejector of compressed air that increases the suction capacity.

The small collectors have tubular filters with interior retention, whereas in the larger ones the filters are flat with exterior retention.

The cleaning of the filters is carried out regularly and automatically with each change of steel or drilling tube. The tubular filters are cleaned with a vibrator that makes them shake, and the flat filters with pneumatic blowing impulses.

The dust can be collected in bags or deposited directly on the bench surface.

2.8 INCLINATION INSTRUMENTS

Recently, a series of devices have been developed, with the common name of inclination instruments, which are used for controlling the direction of the blastholes. Their benefits are:

- Increase in productivity as less time is invested in positioning the feeds.
- Less margin for error in drill alignment which allows larger drilling patterns and deeper blastholes, reducing the powder factor while maintaining fragmentation, and diminishing subdrilling and maintenance costs.

There is a large variety of models that go from the mechanical, through optical and on to the electronic that are the ones most used in underground jobs. Among the most well known are: DIT-70 of Atlas-Copco, Inklinator of Transtonic, Inogon, etc.

2.9 PENETRATION RATE

The penetration rate for a rotary-percussive rig depends upon the following factors:

- Geomechanics, mineralogical characteristics and abrasiveness of the rocks.
- Percussion power of the drill;
- Blasthole diameter;
- Thrust or feed load on the bit;
- Drilling length;
- Flushing of the blasthole;
- Design of the rig and work conditions, and
- Efficiency of the operation.

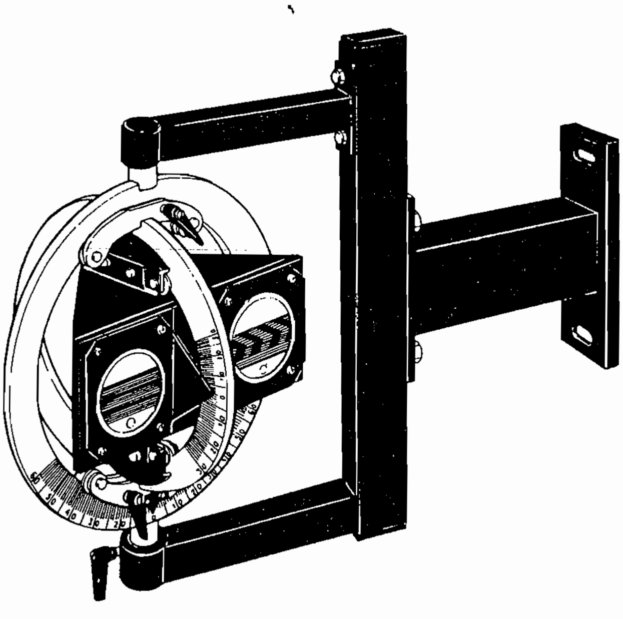


Fig. 2.49. Inclination device.

For a given rig, the penetration rate can be predicted through the following procedures:

- Extrapolation of the data obtained from other work conditions;
- With empirical equations;
- Through laboratory tests taken from representative samples.

The last method is the most precise and trustworthy and will be discussed in more detail.

2.9.1 *Extrapolation of data*

When the penetration rate is known for a given diameter, others can be estimated for the same rig and a smaller or larger diameter by using Table 2.10.

For example, if when drilling at 76 mm an instant penetration rate of 36 m/h is obtained, when at 102 mm the rate will be approximately 36 × 0.65 = 23.4 m/h.

Analytically, the correction coefficient can be calculated from the following equation:

$$F = \left(\frac{D_1}{D_2}\right)^{1.5}$$

2.9.2 *Empirical equations*

One equation that is used to estimate the penetration rate in a rock type, such as Barre Granite of Vermount (USA), and taken as a pattern, is as follows:

$$\mathrm{VP\ (m/min)} = 31\,\frac{\mathrm{POT}}{D^{1.4}}$$

where: POT = Kinetic output power available in hammer (kW), D = Diameter of blasthole (mm).

Therefore, as an example, a hydraulic hammer with a power of 18 kW that is drilling 100 mm diameter blastholes would achieve a penetration rate, in Barre Granite, of 0.88 m/min.

For rocks with a compressive strength above 80 MPa, and when drilling with valveless down the hole hammers, the following equation can be applied:

$$VP = \frac{43 \times p_m^{1/2}\, d_m'^{2}}{RC^{\left(\frac{3.5}{RC}+1\right)} \times D^2 \times D^{1/D}}$$

where: $V =$ Penetration rate (m/h), $p_m =$ Air pressure at hammer entrance (pounds/inches^2), $d_p =$ Piston diameter (inches), $D =$ Blasthole diameter (inches), $RC =$ Uniaxial compressive strength of the rock (pounds/inches^2/100).

Note: 1 pound/inch^2 = 1.423 MPa, 1 inch = 25.4 mm.

Table 2.10. Rate conversions for different diameters.

Blasthole diameter (mm)	127	114	102	89	76	70	64	57	51	48	44	41	38
127	1.00	1.17	1.40	1.71	2.15	2.46	2.83	3.31	3.96	4.35	4.82	5.41	6.10
114	0.85	1.00	1.19	1.45	1.83	2.09	2.41	2.82	3.37	3.71	4.11	4.61	5.19
102	0.72	0.84	1.00	1.22	1.54	1.75	2.02	2.36	2.83	3.11	3.45	3.86	4.35
89	0.59	0.69	0.82	1.00	1.26	1.44	1.65	1.94	2.32	2.55	2.82	3.17	3.56
76	0.46	0.55	0.65	0.79	1.00	1.14	1.31	1.54	1.84	2.02	2.24	2.51	2.82
70	0.41	0.48	0.57	0.70	0.88	1.00	1.15	1.35	1.61	1.77	1.97	2.20	2.48
64	0.35	0.42	0.50	0.61	0.76	0.87	1.00	1.17	1.40	1.54	1.71	1.91	2.15
57	0.30	0.35	0.42	0.52	0.65	0.74	0.85	1.00	1.19	1.31	1.46	1.63	1.84
51	0.25	0.30	0.35	0.43	0.54	0.62	0.72	0.84	1.00	1.10	1.22	1.37	1.54
48	0.23	0.28	0.32	0.39	0.49	0.56	0.65	0.76	0.91	1.00	1.11	1.24	1.40
44	0.21	0.24	0.29	0.35	0.45	0.51	0.59	0.69	0.82	0.90	1.00	1.12	1.26
41	0.19	0.22	0.26	0.32	0.40	0.45	0.52	0.61	0.73	0.81	0.89	1.00	0.08
38	0.16	0.19	0.23	0.28	0.34	0.40	0.46	0.54	0.65	0.72	0.79	0.89	1.00

2.9.3 *Laboratory tests*

A. *Specific Energy Method (US Bureau of Mines)*

The penetration rate is calculated from:

$$VP = \frac{48 \times P_M \times R_e}{\Pi \times D^2 \times E_v}$$

where: VP = Penetration rate (cm/min), P_M = Drill power (kgm/min), R_e = Energy transmission output, usually between 0.6 and 0.8, D = Blasthole diameter (cm), E_v = Specific Energy per unit of volume (kgm/cm^3).

In order to determine the Specific Energy and the Coefficient of Rock Strength *CRS*, it is necessary to carry out a simple laboratory test that consists of letting a weight drop on a rock of about 15 cm^3 a determined number of times and measure the percentage of material under 0.5 mm (Paone and others, 1969). The relationship between the Uniaxial Compressive Strength and the *CRS* is shown in Fig. 2.50.

A Relationship also exists between the *CRS* and the Specific Energy E_v, which is indicated in Fig. 2.51. (Paone, Madson and Bruce, 1969).

B. *Drilling Rate Index (D.R.I.)*

The D.R.I. was developed in 1979, in the University of Tronheim (Norway). In order to obtain the drilling rate, the following tests are carried out with a rock sample of 15 to 20 kg:

– *Brittleness Test.* A representative 500 g fraction cut into 11.2 to 16.0 mm pieces from the rock sample are subjected to 20 successive impacts of a 14 kg weight that is dropped from a height of 25 cm. The process is repeated 3 or 4 times and the mean value is taken from the percentage of samples under 11.2 mm, called the S_{20} value.

– *Drilling test (Sievers' J-value).* With a miniture drill of 8.5 mm in diameter and a 110° bevel angle, subjected to a 20 kg thrust on the rock and making it turn 280 revolutions, 4 to 8 drillings are carried out on each sample. The length of the holes expressed in tenths of milimeters is called the *S-J* value.

The Drilling Rate Index of the rock in question is determined by the S_{20} and *S-J* values from the nomograph of Fig. 2.54.

As can be observed, the D.R.I. coincides with the brittleness S_{20} value when *S-J* is equal to 10, which corresponds to rocks such a granites or the quartzite syenites.

From the data obtained in diverse field investigations, nomographs have been designed where the drilling rates can be estimated with a given hammer, drilling a rock that is characterized by its D.R.I., and working with a determined diameter.

C. *Drillability Index I_p*

This test is carried out in the High Technology School of Mining Engineers, Madrid (E.T.S. de Ingenieros de Minas de Madrid), and its purpose is to reproduce the actual phenomenon of rotary percussion by using an electric drill that slides along a frame and exerts constant thrust on the rock to be studied.

The samples, that are the size of a fist, are prepared by polishing flat surfaces and then setting them in a pot filled with plaster to hold them in place, with the flat surface parallel to the base.

The drill has a diameter of 9.5 mm and 3 or 4 holes are made in 3 to 5 seconds, controlled by an electric temporizer. The dust produced is eliminated by blowing with compressed air.

After testing, the length of the holes are measured with a micrometric probe to obtain their average value. Afterwards, the Drillability Index I_p is calculated like the penetration rate, expressed in inches per minute.

From the investigations carried out by J. Bernaola (1985), in which the actual penetration rates with hammers of different characteristics, different diameters and different types of bits are correlated with the drillability indexes obtained on the same rocks, the output for a given equipment can be predicted by the following steps:

1. Obtention of the drillability index on the rock in laboratory testing.
2. Definition of the drill hammer characteristics: E_g = Energy per blow (pounds × foot), n_g = Number of blows per minute
3. Determination of the length of the cutting edge L_f of the drilling tool. For insert bits the following exists:

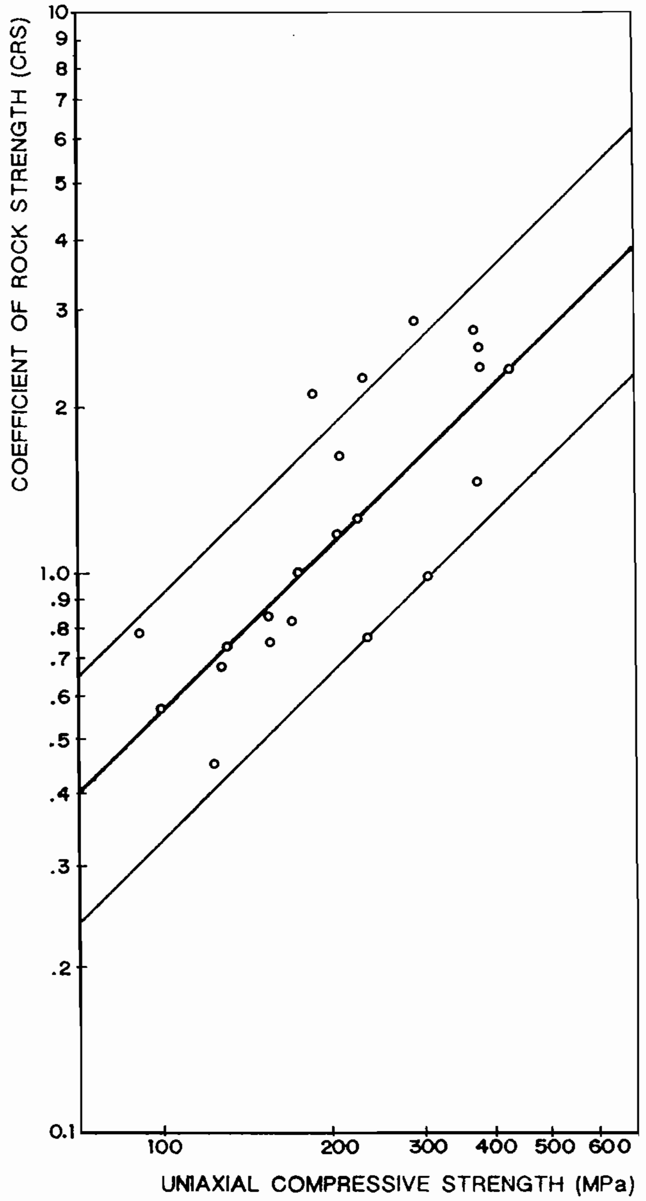


Fig. 2.50. Relationship between the Uniaxial Compressive Strength and the Coefficient of Rock Strength.

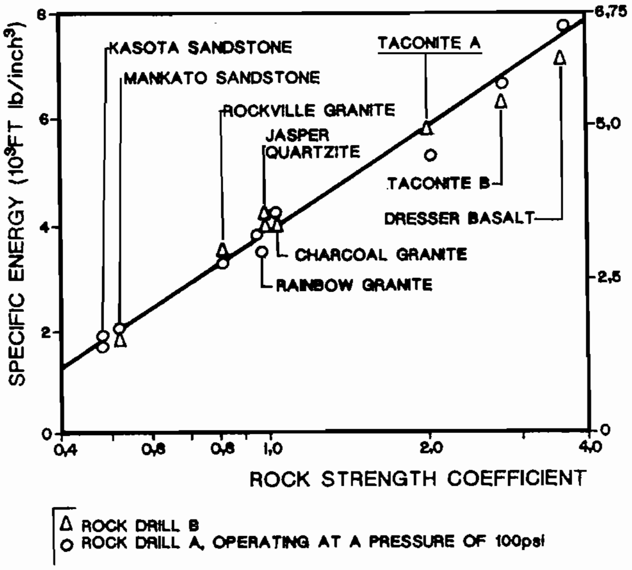


Fig. 2.51. Relationship between the Coefficient of Rock Strength and the Specific Energy.

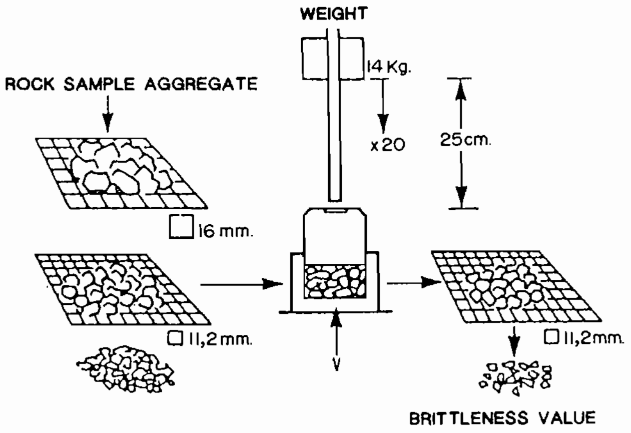


Fig. 2.52. Brittleness test.

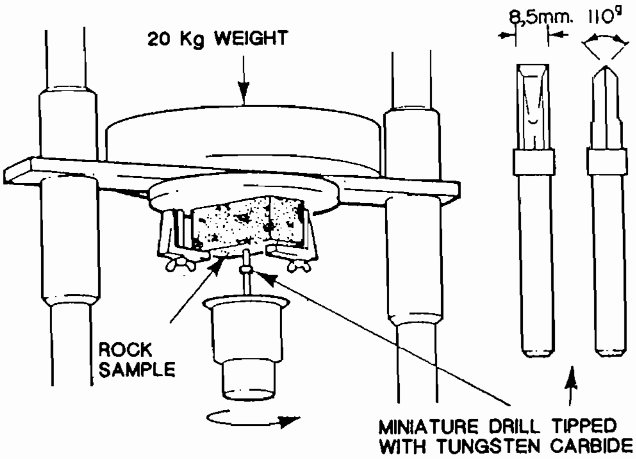


Fig. 2.53. Sievers miniture drill test.

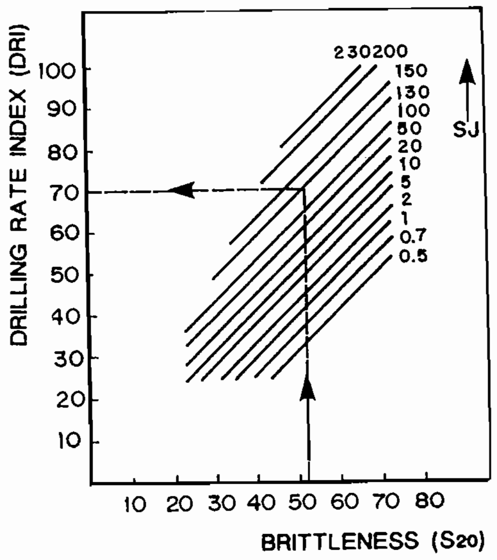


Fig. 2.54. Nomograph for calculating the D.R.I.

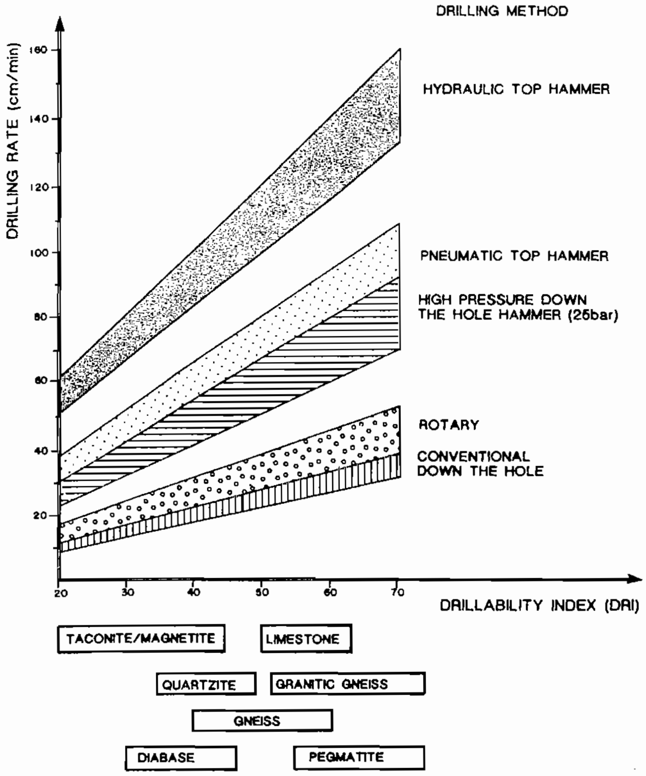


Fig. 2.55. Drilling rates obtained under different work conditions.

$$L_f = 1.7\,D - 0.7$$

where: D = Bit diameter.

If button bits are used, the drilling rate obtained for the studied diameter is multiplied by 1.15, and with bevel bits by 0.85.

4. Calculation of the drilling rate by the formula:

$$VP = \frac{E_g \times n_g}{D \times L_f^{1/2}} \times 10^{-6}(51 \times l_p + 90)$$

2.10 AVERAGE PENETRATION RATE

The average rate achieved by a drill over a long work period depends, apart from the efficiency of the organization, upon the following factors:

- Blasthole depths,
- Maneuvering time.

The lengths of the blastholes are the deciding factor for the number of rods and couplings of the drilling line, which affect the advance rhythms due to energy losses caused by:

– Lack of rigidity in the couplings which give losses of 3% of the energy transmitted by effects of the reflections, and of around 5.5% by frictions that are transformed into heat.

– Inner frictions with the subsequent temperature elevation of the drill steel, due to its transmission of the shock waves. The losses are estimated to be from 0.2 to 0.4 % for each rod.

The figures cited are only valid when working with a top hammer. The resulting mean drilling rate can then be calculated as a function of the number of rods used, taking into account a mean fall in output of 9% which is equivalent to an energy loss of:

$$VP_m = \frac{VP}{N_v} \times \frac{1 - 0.91^{N_v}}{0.09}$$

where: N_v = Number of rods used, VP = Drilling rate obtained with the first rod.

When drilling with down the hole hammers, the drilling rate practically remains constant with the depth because the pipes do not constitute the physical means of percussion energy transmission, as they are only used as air conducts and to carry out rotation.

Once the average value of the drilling rate is obtained, this should be corrected with application of the non-productive times such as:

- Moving the machine from one blasthole to another.
- Positioning and collaring.
- Changing and extraction of the rods.
- Cleaning of the blasthole, jammings, etc.

The following average times exist for surface rigs with and without automatic change, shown in Table 2.11.

The rest of maneuver times are shown in Table 2.12.

Therefore, a rig an with automatic changer, drilling in a low bench that only needs one rod maneuver, has a total non-productive time of 6.9 min.

The previous figures are orientative and can vary in function with the work conditions, rig characteristics, etc.

Another faster way to estimate the final drilling rate consists in using nomographs such as those of Figs. 2.56 and 2.57, which correspond to surface rigs and jumbos, and were made up for pre-established total manoeuvre times.

On the other hand, in the case of full section tunnel and drift driving, it must be taken into account that the cycle lasts one or two shifts, depending upon the section and the degree of support necessary. The total time is usually distributed in the following manner:

- Drilling . . . . 10 – 30%
- Charging of the explosive . . . . 5 – 15%
- Blasting and ventilation . . . . 5 – 10%
- Removal of broken material . . . . 10 – 15%
- Cleaning up and maintenance . . . . 70 – 15%

Table 2.11.

	Change of rods	
	Manual	Automatic
Time to place rod	1.0 min	0.9 min
Time to remove rod	1.5 min	1.0 min
Total rod time	2.5 min	1.9 min

The rest of maneuver times are:

Table 2.12.

Operation	Time
Change of blasthole	3 min
Positioning and collaring	1 min
Blasthole cleaning	1 min

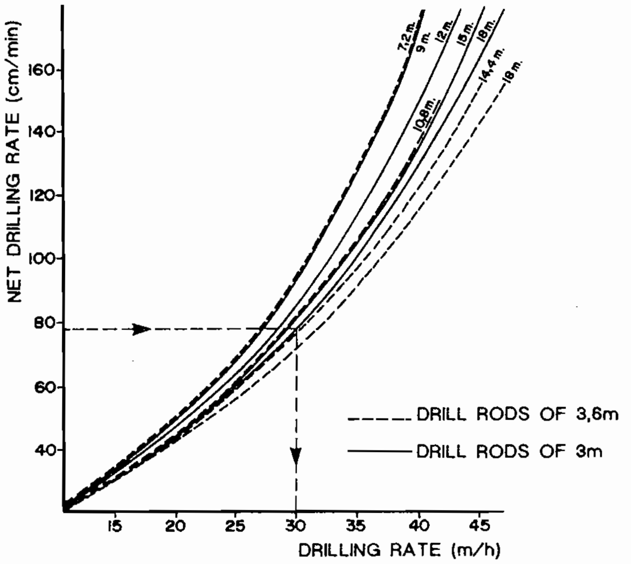


Fig. 2.56. Drilling rates obtained for different bench heights calculating 5 min. time for changing holes and collaring, and 1.9 min. for rod manuevering.

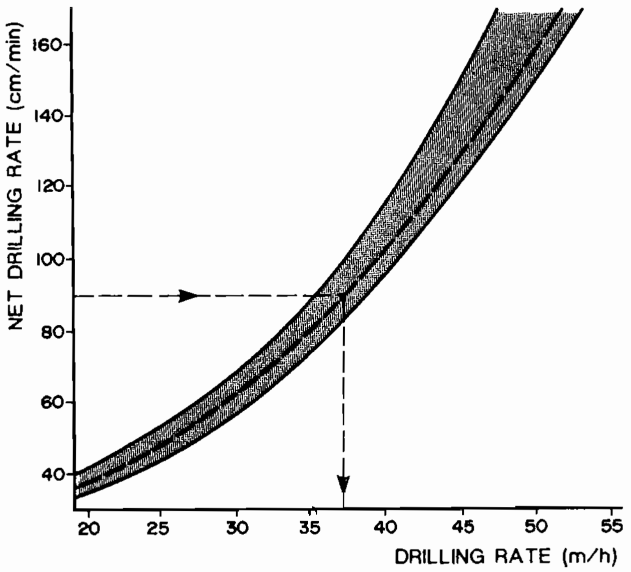


Fig. 2.57. Average drilling rates with mechanized advance for tunnels and drifts.

In the most unfavorable cases the maintenance can take up 70% of the cycle time, which would suggest, in these situations, that a mechanical excavation method should be applied.

Finally, in Table 2.13, the mean data and outputs obtained for different drilling rotary-percussive rigs in a medium type rock are indicated.

2.11 CALCULATION OF DRILLING COSTS

Drilling costs are usually expressed by meter drilled using the following equation:

$$C_T = \frac{C_A + C_I + C_M + C_O + C_E + C_L}{VM} + C_B$$

where:

Indirect Costs. C_A = Depreciation ($/h), C_I = Interest rates and insurance ($/h).

Direct Costs. C_M = Maintenance and repair ($/h), C_O = Labor ($/h), C_E = Fuel or energy ($/h), C_L = Oil, grease and filters ($/h), C_B = Bits, rods, sleeves and shanks ($/h), VM = Mean drilling rate (m/h).

2.11.1 *Depreciation*

Depreciation depends basically on two factors: loss of value and deterioration produced by use and due to aging.

The cost/hour of depreciation, if it considered to be lineal, is calculated in the following manner:

C_A = (Purchase price – residual value) ÷ hours of life

The operative life of the crawler rigs is estimated to be between 8000 and 12.000 hours for those with top hammers, and between 10.000 and 12.000 for those with down the hole hammers. It is important to take into account that the lives of the hammers are probably one half of the indicated figures, so it is wise to include the price of another hammer in the amortization and depreciation figures.

Table 2.13. Data and outputs for rotary-percussive rigs.

	Normal range			Number of operators	Penetration rate (cm/min)	Average drilling rate (m/h)	Compressor	
	Diameter (mm)	Depth					Air flor (l/s)	Pressure (MPa)
		Maxi-mum (m)	Average (m)					
Pneumatic								
Hand held drill 20 kg	32-38	1.5	1.0	1	25	4	30	0.7
Hand held drill 30 kg	38-45	3.0	2.0	1	35	6	60	0.7
Drill wagon on wheels (small)	38-48	8.0	5.0	1-2	45	13	80	0.7
Drill wagon on wheels	48-64	12.0	7.0	1-2	55	16	200	0.7
Drilling rig on crawlers (top hammers)	64-100	20.0	10.0	1-2	60	19	300-350	0.7
Drilling rig on crawlers (down-the-hole-hammer)	85-150	30.0	15.0	1-2	40	13	200	1.2
Hydraulic								
Rig on crawlers (small)	50-75	20.0	10.0	1-2	80	25	70	0.7
Large rig on crawlers	64-125	30.0	15.0	1-2	100	35	80	0.7

Table 2.14.

Equipment (pneumatic)	Repair factor	
	Spare parts only	Parts and labor
Crawler rig without drill		
– For drifter application	4-6%	8-12%
– For down-the-hole application	3-5%	6-10%
Drill for rig use		
– Drifter	6-10%	12-20%
– Down-the-hole	8-12%	16-24%
Hand held drill or drifter	6-10%	12-20%
Portable air compressor	2-3%	4-6%

2.11.2 *Interest rates, insurance and taxes*

To calculate this cost, the following equation is applied:

$$C_I = \frac{\dfrac{N+1}{2N} \times \text{Purchase price} \times \%\,(\text{interest} + \text{insurance} + \text{taxes})}{\text{Work hours per year}}$$

where: N = Number of years of service life.

2.11.3 *Maintenance and repairs*

Include costs of preventative maintenance and breakdowns. It is estimated by the following equation:

$$C_M = [(\text{Purchase price}) \div 1.000] \times FR\,(\%)$$

where: FR = Reparation Factor.

Some orientative figures of the Reparation Factor for pneumatic rigs are shown in Table 2.14, where, on one hand, only the spare parts are considered, and on the other, the parts plus the maintenance labor.

2.11.4 *Labor*

This corresponds to the cost/hour of the driller, including social security, vacations, etc., and of the helper, when necessary.

2.11.5 *Fuel or energy*

This cost is calculated from the specifications of the engines on the machine and the compressor, which can be diesel or electric. For the first, the following equation is used:

$$C_E = 0.3 \times \text{kW POWER} \times \text{FC} \times \text{price of energy}$$

or

$$C_E = 0.22 \times \text{HP POWER} \times \text{FC} \times \text{price of fuel}$$

where: FC = Fuel Factor which varies between 0.65 and 0.85.

2.11.6 *Oil, grease and filters*

This is estimated as a percentage of energy consumption and usually oscillates between 10 and 20%, depending upon the machines.

2.11.7 *Bits, rods, sleeves and shanks*

This is one of the most important aspects that can be calculated from the data indicated in the following chapter, Rotary-Percussive Drilling Accessories.

REFERENCES

Anderson, B.F.: *Down-the-Hole Blasthole Drill Jumbos for Underground Stoping.* Underground Methods Handbook. AIME, 1982.

Arenas, S.: *Sistemas de Perforación para Minería y Construcción. Canteras Y Explotaciones.* Julio, 1985.

Atlas Copco: *Manual Atlas Copco.* 4th edition, 1984.

Auranen, I. *Perforación en Minas a Cielo Abierto. Las Ventajas de la Perforación Hidráulica.* Tamrock News.

Bernaola, J.: *Predicción de la Velocidad de Perforación a Percusión Mediante Ensayo de Laboratorio Sobre Muestras de Roca.* E.T.S. Ingenieros de Minas de Madrid. Tesis Doctoral, 1986.

Bernaola, J.: *Fundamentos de la Perforación Percutiva.* II Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, 1987.

Clark, G.B.: *Principles of Rock Drilling.* Colorado School of Mines. April, 1979.

Clemman, R.J.: *The Design of Percussive Drilling Bits.* Mining & Mineral Engineering. March, 1965.

Frey, G.R.: *Rotary Drills.* Underground Mining Methods Handbook, AIME, 1982.

Gadner Denver: *Rock Drilling Data.* 1986.

Hulkkonem, K. *Nuevas Tendencias en la Perforación de Barrenos Largos.* Tamrock News.

Ingersoll-Rand. *Información Tecnica.*

Kurt, E.H.: *Conventional Small Drilling Equipment.* Underground Mining Methods Handbook, AIME, 1982.

Marshall, D.R.: *Progress in Down the Hole Drilling.* The Quarry Managers Journal. August, 1962.

Menendez, F.: *Perforación a Rotopercusión-Martillo en Cabeza.* II Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, 1987.

Mission: *Percussion Drilling Equipment Operation and Maintenance Manual.* Mission Drilling Products Division. 1985.

Montan, L.: *Recent Developments in Underground Mining Methods and Machinery.* Fifth Australian Tunneling, Sydney, 1984.

Morales, V.: *Perforación Percutiva. Martillos en Fondo.* II Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo. 1987.

Morrel, R.J. & H.F. Unger: *Drilling Machines, Surface.* Mining Engineering Handbook. AIME, 1973.

Patrucco, N.: *Las Perforadoras de Martillo en Fondo. Caracteristicas y Campo de Empleo.* Canteras y Explotaciones. Julio, 1982.

Peach, A.J.: *Increasing Subsurface Excavation Productivity with Computers.* Fifth Australian Tunneling. Sydney, 1984.

Pla, F. et al.: *Curso de Perforación y Voladura.* Fundación Gómez-Pardo. 1978.

Roos, H.H.:*Percussion Drill-Jumbos.* Underground Mining Methods Handbook. AIME, 1982.

Sandvik: *Rock Drilling Manual – Rock Drilling Tool Applications.* 1985.

Stenmark, E.: *Equipos y Métodos Alimak para Minería y Obras Subterráneas.* 1982.

Tamrock: *Handbook of Underground Drilling.* 1983.

Tamrock: *Handbook on Surface Drilling and Blasting.* 1989.

Tandanand, S.: *Principles of Drilling.* Mining Engineering Handbook. AIME, 1973.

CHAPTER 3

Rotary percussive drilling accessories

3.1 INTRODUCTION

In order to carry out a specific drilling job, various combinations of accessories can be chosen. The factors that have to be taken into consideration in the selection are: blasthole diameter and length, structure, strength and abrasiveness of the rocks, size and power of the rock drill, previous experiences and supply facilities.

The extension drill steel is usually made up of the following elements: shank adaptor (1), coupling sleeves (2), extension rods (3) and drill bits (4). Fig. 3.1.

The steels used in the manufacturing of these tools should be fatigue, bending, and impact resistant, and also to wear in threads and shanks.

The ideal is to use steels with a nucleus that is not too hard and a hardened outer surface that is wear resistant. This structure can be obtained in two ways:

a) Steels with a high carbon content, such as those used in shanked rods, including integral drill steel. The desired hardness is achieved by controlling the temperature in the manufacturing process. The part of the shank is treated separately to achieve high impact resistance.

b) Steels with low carbon content. They are used in rods, shank adaptors, coupling sleeves and drill bits. They are steels that contain small amounts of chromium or nickel, manganese and molybdenum.

The treatments that the steels undergo are usually:

– Surface hardening HF (high frequency). Rapid heating to 900°C and fast cooling in water. A high fatigue resistance is obtained and it is used to make rods, coupling sleeves and in some drill bits.

– Carburation. Increase of carbon content on the surface of the steel by introducing the parts during some hours in an oven with a carbon-rich gaseous atmosphere and at a temperature of 925°C. It is used in rods and shank adaptors to achieve high wear resistance.

– Shotpeening with steel shot to increase fatigue resistance in materials that have not undergone the previous treatments.

– Protection against corrosion, by means of phosphatation and application of a fine coating of steel.

The hard metal of the bottons and inserts of the bits is made from tungsten carbide and cobalt by a powder metallurgy technique. This material is characterized by its high wear resistance and durability, and different combinations can be achieved by varying the cobalt content, between 6 and 12%, and the size of the tungsten carbide grains.

The union of the steel with the hard metal can be done by welding in the insert bits and by contraction or pressure in the case of button bits.

3.2 TYPES OF THREADS

The function of drill steel threads is to hold the shanks, coupling sleeves, rods and bits together during drilling. They should be efficiently adjusted so that the elements of the extension rod are tightly united in order to achieve direct transmission of energy. However, they should not be over tightened because this would make uncoupling of the rods more difficult when removed from the blasthole.

The characteristics that determine if the drill rods are easily uncoupled or not are: the angle of the profile and the pitch of the thread. A larger pitch combined with a smaller angle of profile will make the thread easier to remove, in comparison with threads of the same diameter.

The principle types of threads are:

– *R-thread.* Used in small blastholes with drill rods of 22 to 38 mm and powerful independent rotation rock drills with air flushing. It has a pitch of 12.7 mm and a large profile angle.

– *T-thread.* It is adequate for almost all drilling conditions and is used with drill rod diameters of 38 to 51 mm. It has a larger pitch and a smaller slope angle which makes decoupling easier than in the R-thread, and also one of the sides has a large volume for wear which gives it long life.

– *C-thread.* Designed for large extension rods of 51 mm. It has a large pitch and a slope angle that is similar to that of the T-thread.

– *GD or HL-thread.* This thread has intermediate characteristics between the R and T threads. It has an asymmetrical 'sawtooth' profile and is used in 25 to 57 mm rod diameters.

When drilling certain soft rocks, the threads may have double length, so that the first part can be cut off when worn out and work can be continued with the second.

There are special threads, such as the spiral thread over the whole length of the rod. The reason is also for more wear, as when an end wears out it can be cut off, but the problem is that standard lengths are not available. The diameters of these rods come in 32, 38 and 45 mm.

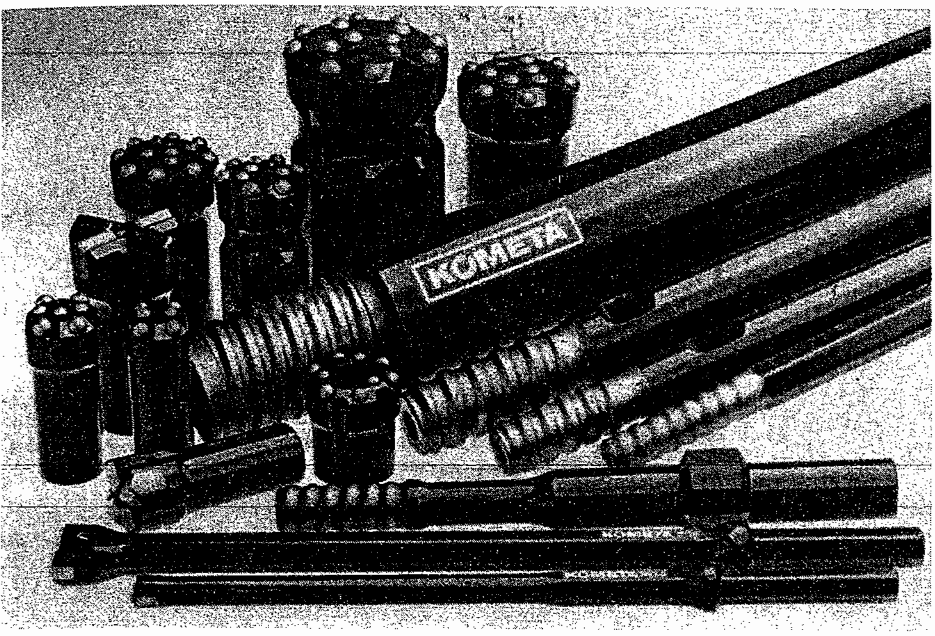


Photo 3.1. Drilling accessories (Courtesy of Kometa).

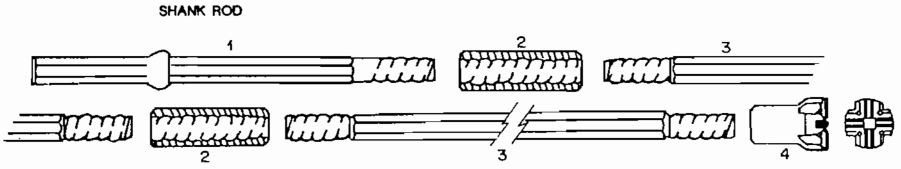


Fig. 3.1. Extension drill steel equipment.

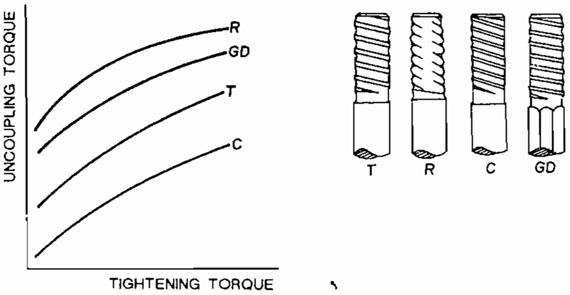


Fig. 3.2. Types of threads (Atlas Copco).

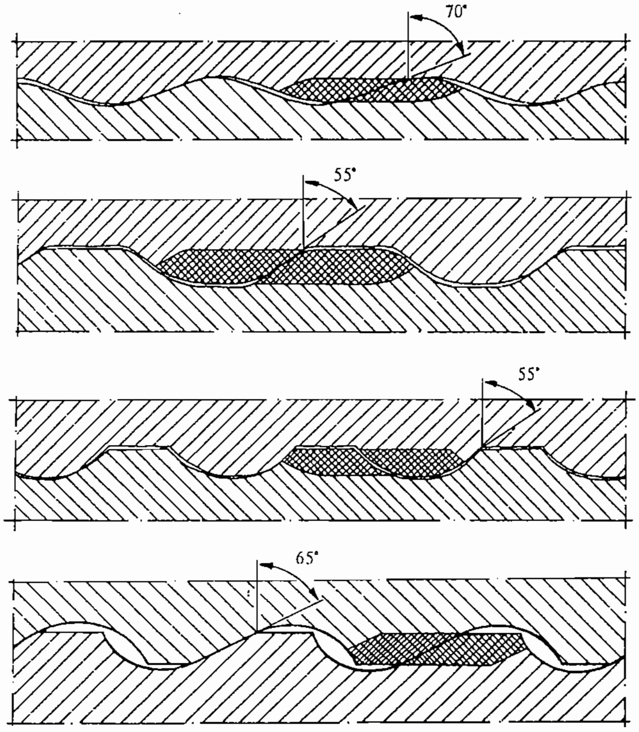


Fig. 3.3. Profiles of the R, T, C and GD threads.

3.3 SHANK ADAPTORS

Shank adaptors are those elements that are fixed to the rock drill chuck in order to transmit impact energy to the rods, rotation torque and feed force. There are two types of shank adaptors 1) The Leyner shank and 2) The splined shank, Fig. 3.4.

The first type is used with extension rods of 25 and 32 mm, whereas the splined shanks, of 4 to 8 splines, are used with 38, 44 and 50 mm diameters, with independent rotation hammers. In modern rock drills with an impact power of at least 18 kW, the shank adaptors are designed without a 'slimmed' zone between the chuck section and the screwed end, thereby reinforcing the impact surface.

The flushing system can be through a flushing tube in the rock drill, in which case the shank adaptors have an inner sealing which is the part that comes into contact with the flushing water, or lateral, Fig. 3.5, with an opening between the splines and the thread where the flushing fluid enters through a separate swivel device that is concentrically joined with the shank adaptor.

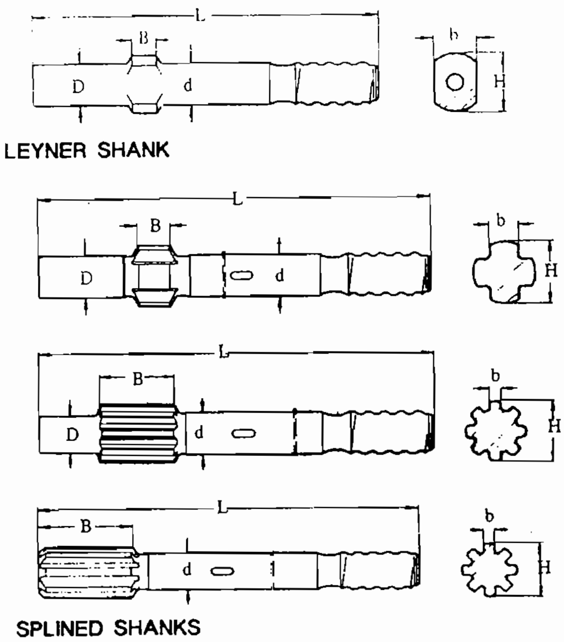


Fig. 3.4. Shank adaptors.

Another design, that can be seen below, consists of drill rods that have a shank at one end. Fig. 3.6. They are used in handheld drills or with small diameter hammers, of 19, 22 and 25 mm and have a hexagonal transversal cross-section.

3.4 DRILL STEEL

The parts that extend from the drill chuck are usually:
- Rods or bars,
- Pipes.

The first are used when drilling with top hammer and can have a hexagonal or round cross-section. The extension rods have external male threads and are joined together with coupling sleeves.

Extension rods of hexagonal complete section a) or round b), Fig. 3.7, have the same size in the center of the rod as in the threads. In the first, the hexagon circumscribes the circle which corresponds to the similar ones of round cross-section and, for this reason, they are more rigid and a bit heavier.

When the drilling conditions are such that the lives of the rods only depend upon the wear of the threads, double length threads are used c). Thus, when the first part of the thread is worn, drilling can still be carried out with the second part.

The light extension rods d) have transversal cross-sections, usually hexagonal, that are smaller than those of the thread. The selection of this type of rod refers to the thread sizes.

Recently, rods with integral coupling sleeves e) have appeared on the market, with easier handling as they eliminate the use of sleeves, energy transmission is better, the blastholes are straighter and the operation is safer. The price of these rods is the same as that of a conventional

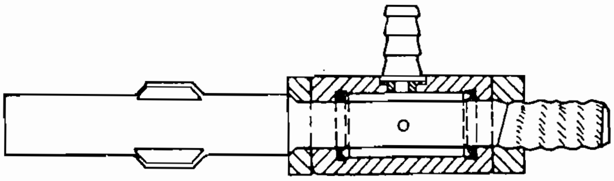


Fig. 3.5. External swivel time.

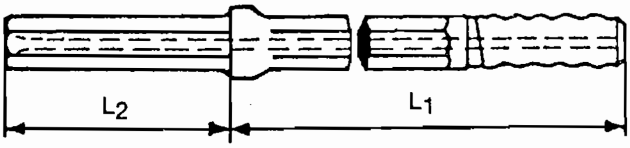
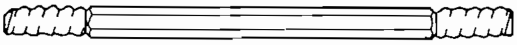
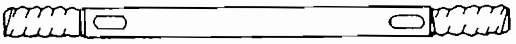


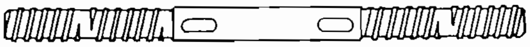
Fig. 3.6. Shank rod.



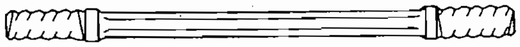
a) HEXAGONAL



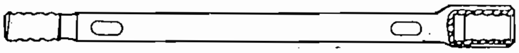
b) ROUND



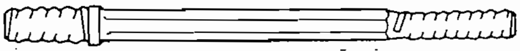
c) DOUBLE THREAD



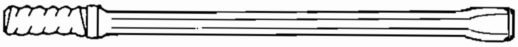
d) LIGHT



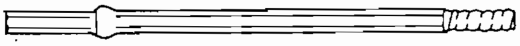
e) WITH INTEGRAL COUPLING SLEEVE



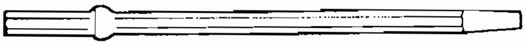
f) EXTENSION ROD FOR DRIFTING AND TUNNELING



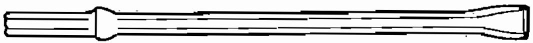
g) THREAD ROD WITH INTEGRAL JOINT



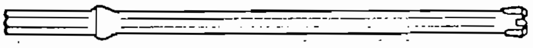
h) SHANKED ROD WITH THREADED JOINT



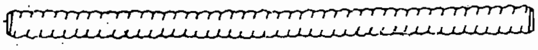
i) SHANKED ROD WITH TAPERED JOINT



j) SHANKED ROD WITH INTEGRAL JOINT



k) SHANKED ROD WITH INTEGRAL JOINT AND THREE BLADES



l) CONTINOUS THREAD

Fig. 3.7. Types of rods.

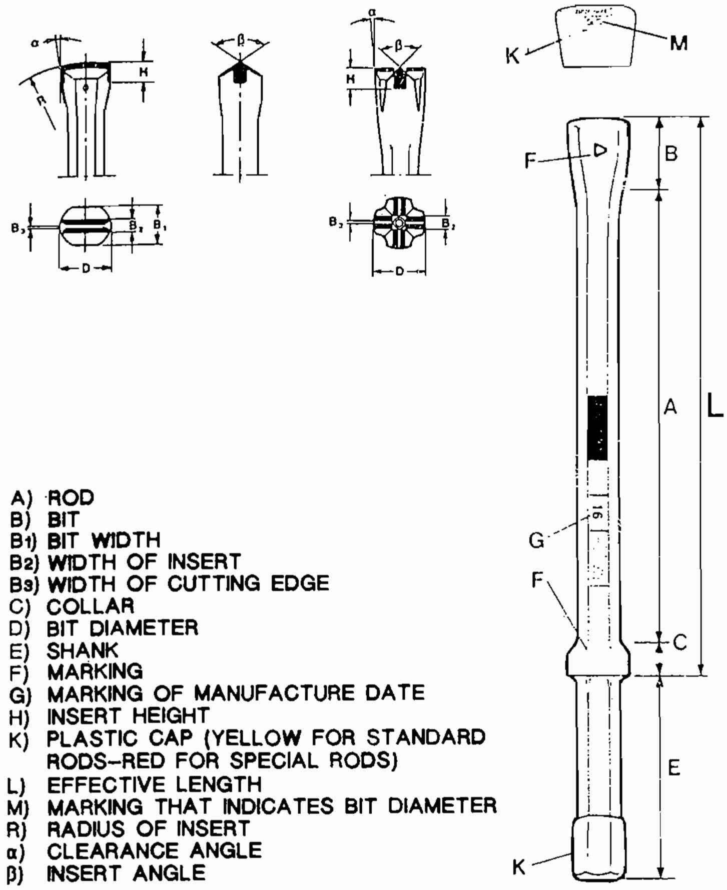


Fig. 3.8. Integral drills (Sandvik-Coromant).

rod plus the sleeve, but if they break at the points of union they become useless.

There are various types of drill steel for tunneling, among which the following should be mentioned: the extension rods for drifting and tunneling f), which have a larger diameter thread at one end than that of the central cross-section. The size chosen refers to the size of the thread at the shank end.

The thread rods with integral joints g), of hexagonal cross-section, that have an insert bit at one end and thread at the other. The shanked rods with threaded joints h) which have a hexagonal shank at one end and thread at the other, and the shanked rods with tapered joints i).

Lastly, there is the group of integral drill steels with a forged shank at one end and a forged bit with cemented carbide inserts at the other, Fig. 3.8, that are subdivided according to the shape of the bit and the shape of the inserts.

The integral drill steel sets are arranged in series in which the hole diameter decreases with increasing steel length. The main types are:

– *Chisel steel.* These are the most commonly used because of their low cost and easy sharpening.

– *Multiple insert steel.* They are used in the mechanized drilling of soft and fissured rocks.

– *Button steel.* Used in rocks that are easy to drill and with little abrasiveness, such as coal.

– Steel for working with marble, with four blades and special channels for the evacuation of drill cuttings.

In surface drilling, the hexagonal rods are commonly used with light rigs and manual change, whereas those of round section are used when the rock drills have automatic rod changers.

In Table 3.1. the available diameters and standard lengths of the most common types of rods are indicated.

On the other hand, in Table 3.2. the drill steel diameters and maximum lengths drilled in different blastholes are shown.

With the application of top hammer hydraulic rock drills to the drilling of large diameter blastholes, above 115 mm, some drill pipes, or tube rods, have been recently designed which are similar to those used in down the hole hammer operations.

The main advantages of this tubular drill steel are:

1. More rigidity. The deviations and irregular blasthole walls are reduced, as they have larger diameters (76 to 165 mm).

2. Improved energy transmission as couplings are not necessary.

3. More efficient flushing by improving the upward air velocity outside the tubes and by being able to increase the amount of pumped air.

Table 3.1.

Extension rod	Diameters available	Lengths available
Type		
Hexagonal, normal	25, 28, 32, 38 mm	3050, 3660 mm
Round, normal	32, 38, 45, 51 mm	3050, 3660, 6100 mm
Round, MF	32, 38, 45, 51 mm	3050, 3660, 6100 mm

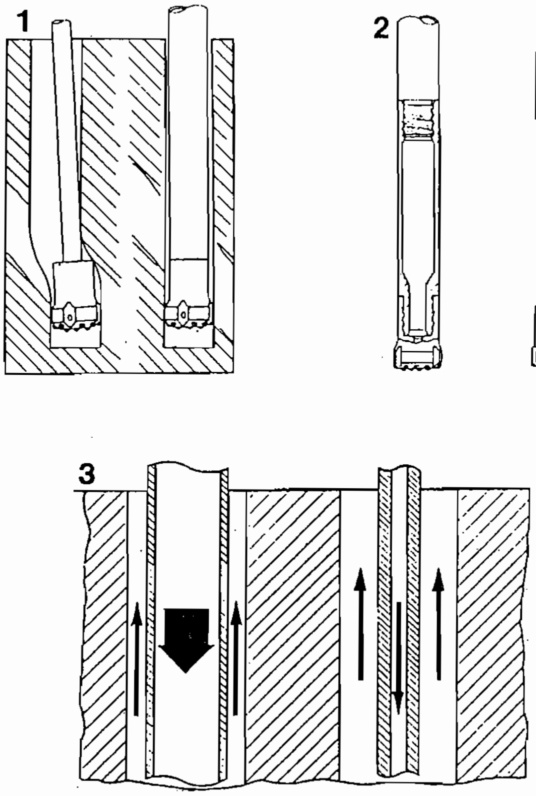


Fig. 3.9. Drill pipes (Tamrock).

Table 3.2.

Rod diameter (mm)	(in)	Bit diameters (mm)	Recommended maximum hole length (m)
25	1	38, 41, 45, 51	6...8 m
28	1⅛	38, 41, 45, 51	8...10 m
32	1¼	48, 51, 57, 64, 76	12...15 m
38	1½	64, 70, 76, 89, 102	15...18 m
45	1¾	76, 89, 102, 115	18...22 m
51	2	89, 102, 115, 127	25...28 m

Table 3.3.

Pipe diameter (mm)	Length (mm)	Thread API reg.	Weight (kg)
76	1500	2⅜"	15
76	3000	2⅜"	25
89	1500	2⅜"	22
89	3000	2⅜"	44
89	4500	2⅜"	63
114	1500	3½"	45
114	3000	3½"	61
114	6100	3½"	170
115	7600	3½"	199
127	6100	3½"	204

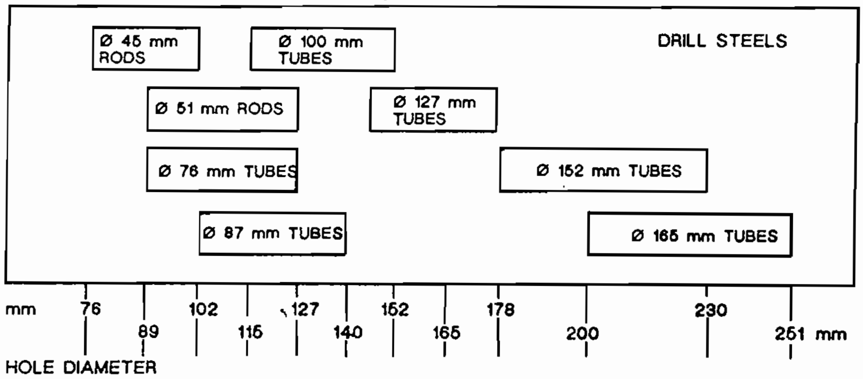


Fig. 3.10. Recommended tube drill steels for different drilling diameters.

Fig. 3.10 gives the recommended drill pipe diameters in function with blasthole size.

There are also guide pipes on the market which have one or two cross sections at each end with four longitudinal external blades. They are manufactured with male-female threads at each end, which eliminates the need for couplings. These pipes allow the drilling to be carried out with deviations of less than 1% and adequate for surface as well as underground drilling.

The guide pipes are placed behind the drill bit, giving additional support. the rest of the support is made up of rods with 45 to 51 mm diameters. As the guide pipe is at the bottom of the blasthole, it produces an effect similar to that of a complete pipe string.

Finally, when drilling with down the hole hammers, pipes, as indicated, are used with lengths of 3 to 7.5 m and male/female threads on the ends. There are notches near the threads which facilitate connecting and unthreading the pipes.

The standard sizes, for each pipe diameter, and its approximate weight are indicated in Table 3.3.

3.5 COUPLINGS

Couplings are to connect the extension rods to each other until the desired length is reached, with sufficient adjustment to assure that the ends meet so that energy transmission is effective.

The couplings that are available are as follows:

a) Sleeve or through type,
b) Semi-bridge type,
c) Full-bridge type
d) With helical splines on the outside.

Table 3.4.

Bit diameter		Largest rod diameter		Coupling diameter	
(mm)	(in)	(mm)	(in)	(mm)	(in)
41	$1\frac{5}{8}$	25	1	36	$1\frac{7}{16}$
45	$1\frac{3}{4}$	28	$1\frac{1}{8}$	40	$1\frac{5}{8}$
51	2	32	$1\frac{1}{4}$	44	$1\frac{3}{4}$
57	$2\frac{1}{4}$	32	$1\frac{1}{4}$	44	$1\frac{3}{4}$
64	$2\frac{1}{2}$	38	$1\frac{1}{2}$	55	$2\frac{5}{32}$
70	$2\frac{3}{4}$	38	$1\frac{1}{2}$	55	$2\frac{5}{32}$
76	3	45	$1\frac{3}{4}$	63	$2\frac{31}{64}$
89	$3\frac{1}{2}$	51	2	72	$2\frac{7}{8}$

e) With large diameter blades.

The couplings with a central stop b) and c) keeps them from slipping along the drill steel. They are used in all T-threads and at the end of the shank on drill steels for tunneling and drifting.

The spline type d) are used with retract bits in blastholes with jamming tendencies. The couplings with blades serve to centralize and stabilize the rods.

The thermic treatments during manufacturing are surface hardening, total carburation or only on the inside.

Coupling diameters for different size drill steel are indicated in Table 3.4.

3.6 DRILL BITS

There are two types of drill bits for rotary percussive drilling:

- Insert bits, and
- Button bits.

Some of the design characteristics that the two types of bits have in common are the following:

– The rods are threaded to the end of the bit thread so that the transmission of the impact energy is as direct as possible on the rock.

– The bits have a series of central and lateral openings through which the flushing fluid is injected and they have channels through which the rock particles produced pass upwards.

– The bits are designed to be slightly conic, with the widest part in contact with the rock so as to counteract the wear and avoid an excessive adaptation to the blasthole wall.

a) *Button bits*

These bits have buttons or cylindrical inserts of tungsten carbide distributed in various pattern on the face. They are manufactured in diameters that go from 50 mm up to 251 mm.

Button bits are better adapted to rotary drilling, obtaining advance velocities above those of insert bits. They are also more wear resistant, due not only to the shape of the buttons but also to the more effective attachment of the buttons to the steel, by shrinking or cold-pressing.

b) *Insert bits*

They have two design configurations: 1) cross bits and 2) X-bits. The first consist of four tungsten carbide inserts at right angles to each other, whereas the X-bits have four inserts at angles of 75° and 10° to each other.

These bits are manufactured from 35 mm diameters up, being normal to reach 57 mm in cross bits and use X-bits from 64 mm on as they are faster and avoid the tendency of the others to open large diameter blastholes with pentagonal cross-sections.

c) *Special bits*

Bits with special design are known as:

- Retrac bits,
- Reaming bits,
- Drop center bits, and
- Ballistic bits.

The retrac bits are used in rock formations where the walls tend to collapse and, therefore, it is necessary to avoid jammings and loss of drill steel. They have splines and cutting edges cut out of the bit wing which allow them to drill during extraction.

A variation of this bit is the retrac bit with a long skirt. With this utensil, the reverse cutting is more intense, and its constant diameter achieves straighter blastholes.

The reaming button bits are used for underground labors to open the central blastholes of larger diameter in parallel cuts. These bits are used with pilot rods or extension rods and reaming bit adaptors. They have a central conical opening which allows them to be placed behind the rod with the smallest diameter.

The drop center bits have excellent flushing characteristics, as the flushing hole of the bit is in the center of the face. They are used in soft rocks that are easy to drill. These accessories also improve the straightness of the blastholes.

The ballistic bits have bullet shaped buttons which are longer than the standard and give higher penetration rates and a more efficient flushing. In soft rocks, the face of the bit does not strike upon the rock in the bottom of the blasthole, owing to the button height, which makes the cleaning out of drill cuttings more complete. In comparison to standard button bits, the ballistic bits give higher drilling rates, from 25 to 50% more, depending upon the type of rock.

The main problem is the risk of button breakage, especially when the body of the bit is subjected to more wear than the buttons themselves.

In Table 3.5, the types of bits recommended for drilling different rock formations are indicated.

d) *Down the hole hammer bits*

The bits for down the hole hammers have shanks incorporated upon which the pistons strike directly. The most common diameters of these bits go from 85 mm to 250 mm, although larger ones exist.

The main types of bits are the following:

Button bits. These are the most commonly used and are good for any type of rock. They are subdivided into: Bits with a breaking nucleus, concave bits, convex bits, ballistic bits.

Insert bits. With a complete face. With cross or X inserts, similar to those of the top hammer and for application in soft or loose rocks. With a breaking nucleus. Bits with four short inserts and one or two buttons in the center that serve to break the nucleus of rock that forms in each blow.

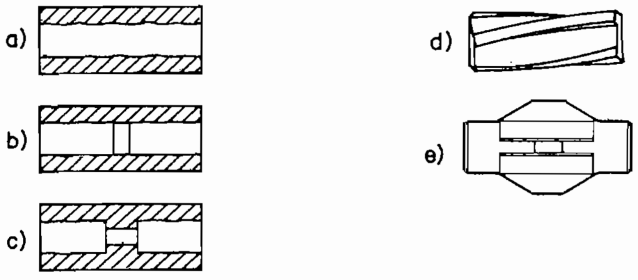
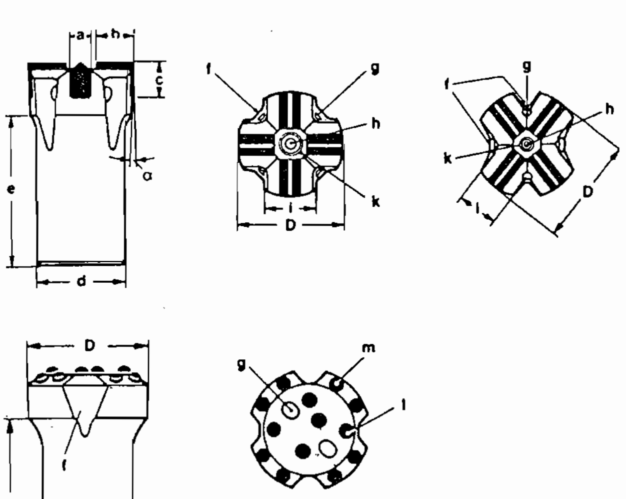


Fig. 3.11. Types of couplings.



a) INSERT WIDTH
b) INSERT LENGTH
c) INSERT HEIGHT
d) SKIRT DIAMETER
e) SKIRT LENGTH
f) SLUDGE GROOVE
g) SIDE FLUSHING HOLE
h) CENTRAL FLUSHING HOLE
i) WIDTH OF FLANK
k) CENTER METAL INSERT
l) CENTER BUTTON
m) GAUGE BUTTON
D) BIT DIAMETER
α) CLEARENCE ANGLE

Fig. 3.12. Drill bits (Sandvik-Coromant).

Table 3.5.

Rock properties	Insert bits Nor-mal	Heavy duty	Retrac	Button bits Nor-mal	Heavy duty	Retrac
Soft	R	N	N	R	N	N
Medium-hard	S	R	N	R	S	N
Hard	N	S	N	S	R	N
Heavy gauge wear	N	R	N	N	S	N
Heavy frontal wear	N	S	N	S	R	N
Moderate frontal wear	N	N	N	R	S	N
Fissured rock	N	N	R	N	N	R

R = Recommended, S = Suitable, N = Not recommended.

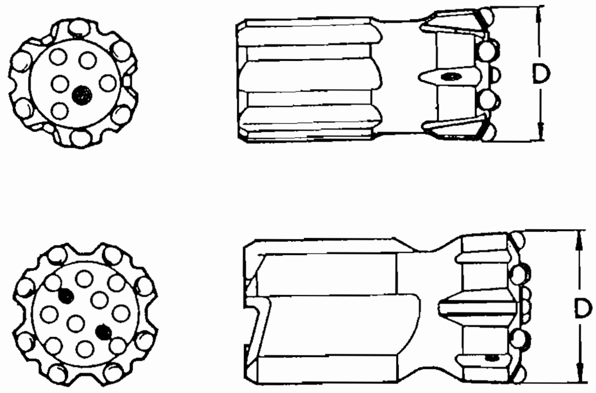


Fig. 3.13. Retrac bits of button type.

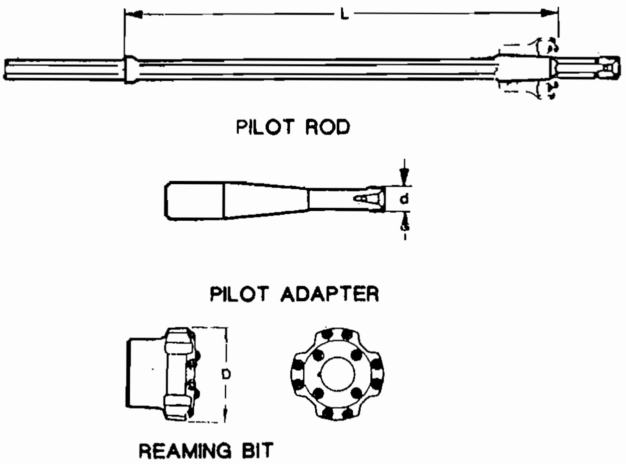


Fig. 3.14. Reaming bit, extension rod and bit adaptor.

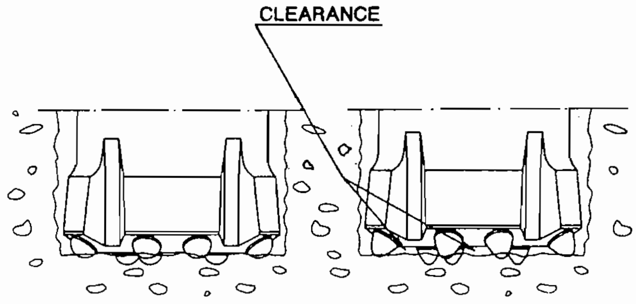


Fig. 3.15. Conventional bit and ballistic bit.

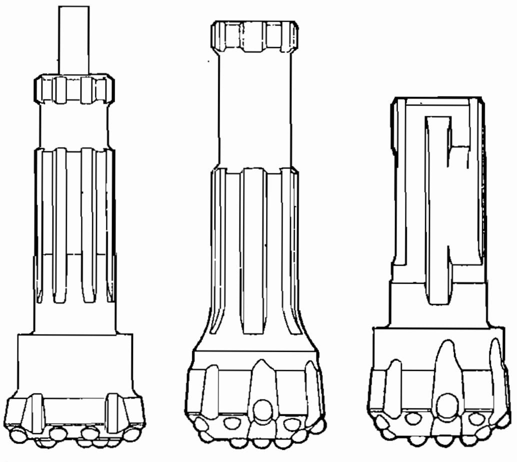


Fig. 3.16. Down the hole hammer bits.

3.7 CALCULATION OF THE NECESSARY DRILLING ACCESSORIES

The amount of drill steel that is necessary to carry out a job depends upon diverse factors:

- The volume of rock.
- The specific drilling.
- Drillability and abrasiveness of the rock, and
- Drilling method.

The service life of the drill steel is basically decided by the last two factors and, above all by the drillability in abrasive rocks. Frequently the life of these accessories is expressed in 'rod-meters', owing to the fact that the number of meters drilled with a given rod is in function with its length and the blasthole depth.

Example:
Length of blasthole = 12 m.
Length of extension rod = 3.05 m.

$$\frac{\text{Total meters drilled}}{\text{Total meters-rod}} = \frac{30 \text{ m-v}}{12 \text{ m}} = 2.5 \text{ m-v/m}$$

When the length of the rod is 3 m, then the mean value is 7.5 rod-meters for the blasthole of the indicated depth.

Generally, the following exists:

$$MV = L \times \left[\frac{L + L_v}{2L_v}\right]$$

where: L = Length of blasthole (m), L_v = Length of each extension rod (m), MV = Rod-meters.

In order to estimate the drilling accessories required for a given project, the following equations can be used:

1. Number of bits

$$N_B = \frac{VR \times PS}{V_B}$$

2. Number of extension rods

$$N_V = \frac{VR \times PS}{V_V} \times \frac{L + L_V}{2L_V}$$

3. Number of shanks

$$N_A = N_V/3$$

4. Number of couplings

$$N_M = 1.5 \times N_V$$

where: VR = Volume of rock to be blasted (m^3), PS = Specific drilling (ml/m^3), L = Blasthole depth (m), V_i = Service life of each accessory.

To give an idea, the lives of the different types of bits can be estimated for diverse bench drilling operations and for tunnel and drift driving from Tables 3.6 and 3.7.

The life expectancy of drill steel can be determined by knowing the following factors:

- Type and size of the threads.
- Number (N_v) and length of the extension rods (L_V) required to drill a hole with a length (L).
- Penetration rate (*VP*), which depends upon the type of rock, drilling diameter and type of hammer, Fig. 3.17.

The life of the couplings is considered to be equal to that of the drill steel, although they usually last somewhat less.

Table 3.6. Service life of accessories in bench blasting.

Accessory	Type of rock Abrasive	Slightly abrasive
Integral drill steels		
Regrinding interval	20-25	150
Service life	150-200	600-800
Threaded insert bits		
Regrinding interval	20-25	150
Service life	200-400	800-1200
Threaded button bits		
– Diameter ≥ 64 mm		
Regrinding interval	60-100	300
Service life	400-1000	1200-2500
– Diameter < 57 mm		
Regrinding interval	100-150	300
Service life	300-600	900-1300
Down-the-hole button bits		
Regrinding interval	40-60	300
Service life	400-1000	1200-2500
Extension rods		
Service life	600-1800	
Coupling sleeves		
Service life	100% life of the rods	
Shank adaptors		
Service life		
– Pneumatic rock drills	1500-2000	
– Hydraulic rock drills	3000-4000	

Note: The figures are in metres. Source: Atlas Copco.

Table 3.7. Service life of accessories in tunnels and drifts.

Accessory	Type of rock Abrasive	Slightly abrasive
Integral drill steels		
Regrinding interval	20-25	150
Service life	200-300	700-800
Threaded insert bits		
Regrinding interval	20-25	150
Service life	250-350	900-1200
Threaded button bits		
Service life	250-550	1000-1300
Extension rods		
Service life		
– Pneumatic rock drills	1000-1500	
– Hydraulic rock drills	1600-1400	
Threaded integral steels		
Service life	600-800	
Coupling sleeves		
Service life	100% of rod service life	
Shank adaptors		
Service life		
– Pneumatic rock drills	1200-1600	
– Hydraulic rock drills	2500-3500	

Note: The figures are in metres. Source: Atlas Copco.

3.8 CARE AND MAINTENANCE OF THE BITS

The object of bit conditioning is to optimize the drilling rate and increase service life.

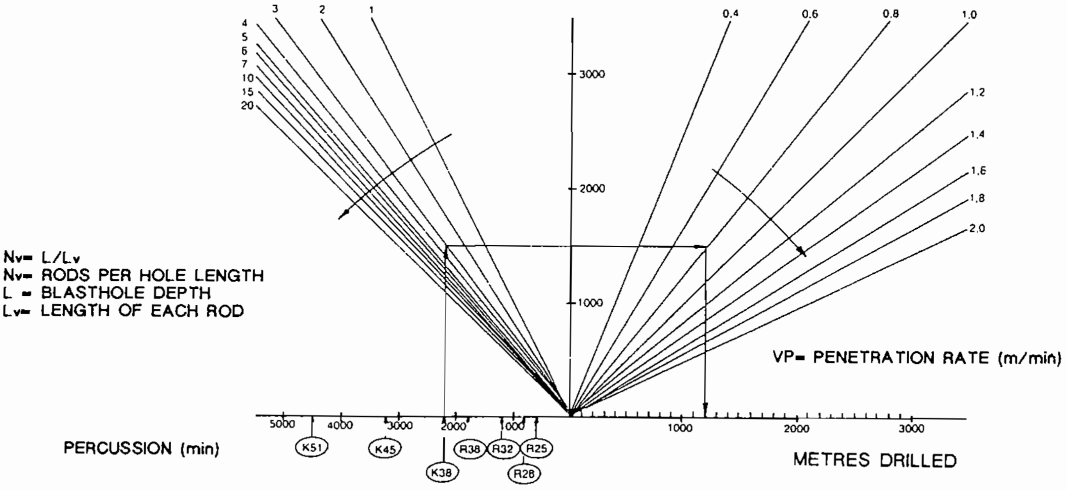


Fig. 3.17. Example of service life calculation for drill steel (Kometa).

It is only logical that if the hard metal inserts or buttons and the rest of the body of the bit do not have an adequate shape then the highest drilling rate will not be achieved and, apart from this, tensions and stresses will be generated in the tool itself as well as in the rest of the drill steel with the consequence of possible damage or breakage.

The next paragraphs will indicate when and how the resharpening should be carried out for button and insert bits, as well as for integral drill steels.

a) *Button bits*

Button bits should be reconditioned when:

1. The body of the bit is more worn than the buttons, making these stand out excessively. This will keep the buttons from getting stuck in the rock or shattered, which happens quite often in soft and abrasive ground, Fig. 3.18.
2. When the buttons wear flat more rapidly than the body, especially in hard and abrasive rocks, they should be sharpened frequently, Fig. 3.19.
3. If, in non abrasive rock the buttons are worn smooth and show signs of surface cracking, called snake skin. This keeps the cracks from spreading which could destroy the buttons, Fig. 3.20.

The sharpening or regrinding of buttons should return their original spherical shape, with as little loss of height as possible. It is usually not necessary to grind the circumference.

The grinding intervals can be chosen in function with the different types of rocks and drilling conditions, for example, after a certain number of blastholes, or when almost half of the button diameter is worn away, Fig. 3.21.

If the bits are very worn, it might be necessary to grind down the steel around the buttons so that they maintain their height. The visible height should be about half of the diameter.

All the buttons should be reground at the same time, even though they are not all worn to the limit. The bits can be used for drilling whenever the peripherical or gauge buttons are in good condition, as they are the most important. Special attention should be paid to cleaning the openings and the flushing channels.

Buttons should be sharpened with grinding wheels and controlled with templates.

b) *Insert bits*

Insert bits should be reground when:

1. The cutting edge of the bit has worn flat to a width of 2.4 mm at 5 mm distance from the periphery of the bit, Fig. 3.22.
2. When the outer corner of the insert is worn to a radius larger than 5 mm, Fig. 3.23.
3. When the bit face begins to have a diameter that is smaller than that of the body; the periphery should be reground to eliminate anti-taper, Fig. 3.24.
4. In non-abrasive ground where the inserts show smooth worn areas or surface micro-cracks, which must eliminated periodically, Fig. 3.25.

The sharpening of this type of bits should be carried out in such a manner that the mean insert angle be 110° and the angle of the body of around 3°, Fig. 3.26.

The corners of the inserts should not be sharpened but left with a slight chamfer. A wedge shape of the inserts should be avoided. A slightly convex shape is recommended with a maximum angle of 10 to 15%, Fig. 3.27.

If the grinding is done without a stream of water, the bits should be cooled slowly with air before continuing the regrinding process. The cutting edges of the inserts, once the bits have been ground, should be chamfered until they reach a wear flat of 0.4 to 0.8 mm, Fig. 3.28.

If the body of the bit has been worn, the part of the inserts that stand out should be ground down until they are flush with the body. The flushing channels should also be conditioned and the bits oiled after sharpening and before reuse.

c) *Integral drill steel*

These accessories should be sharpened when the wear flat reaches 3 mm width, measures a 5 mm from the outer edge. In abrasive rocks or drilling with air, the edges that have become rounded should also be sharpened and given a conical shape, up to a height of 8 mm, Fig. 3.29.

The geometry that should be reached in grinding is a cutting edge angle of 110° and a curving of 80 to 100 mm, Fig. 3.30.

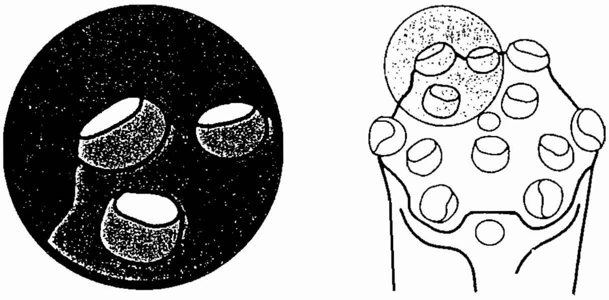


Fig. 3.18. Body wear.

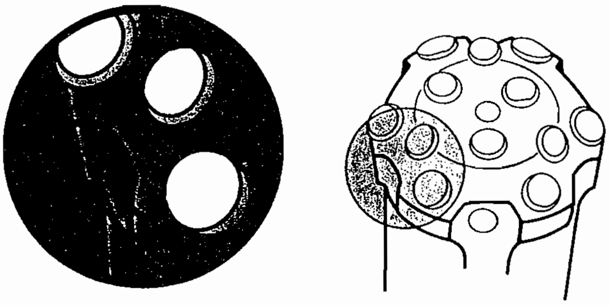


Fig. 3.19. Wear flat of the buttons.

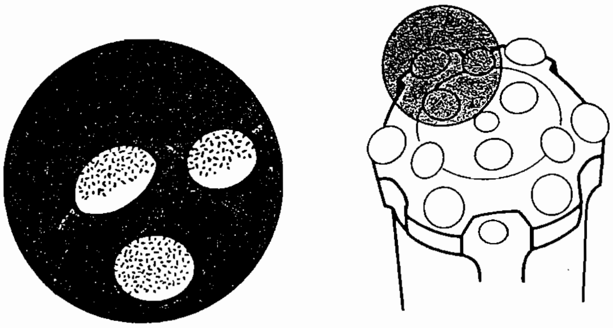


Fig. 3.20. Buttons worn smooth with snake skin aspect.

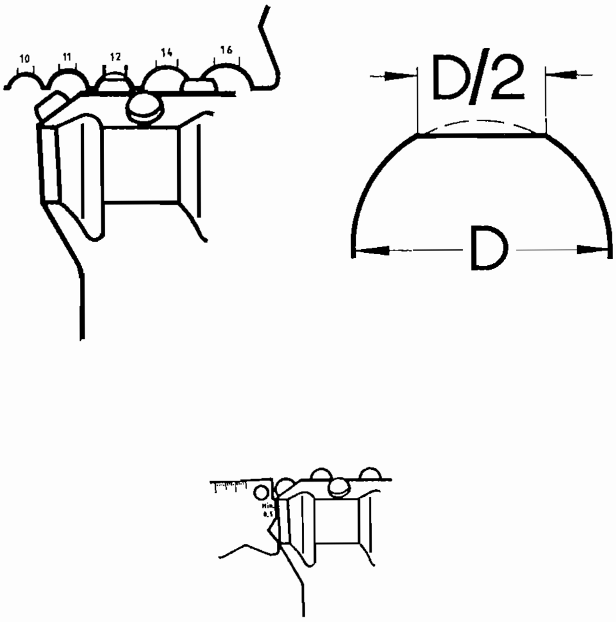


Fig. 3.21. Measurements of button wear.

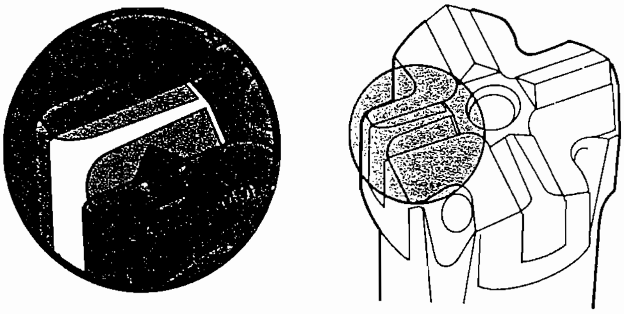


Fig. 3.22. Wear flat of the cutting edge of the inserts.

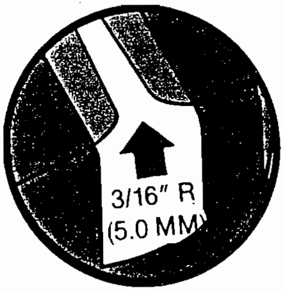


Fig. 3.23. Wear of the corners.

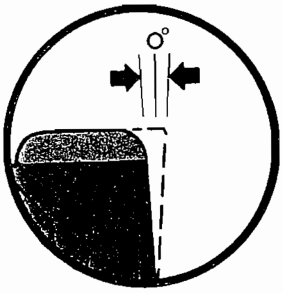


Fig. 3.24. Anti-taper.

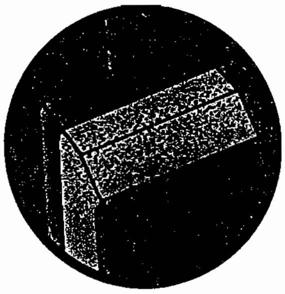


Fig. 3.25. Inserts worn smooth.

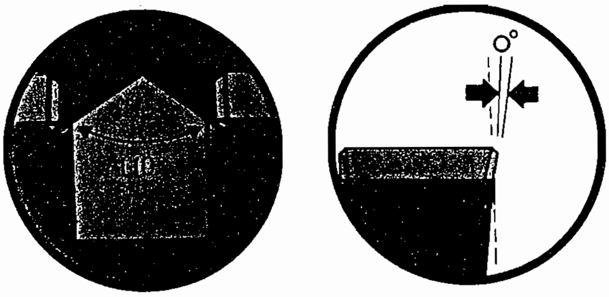


Fig. 3.26. Sharpening of the inserts.

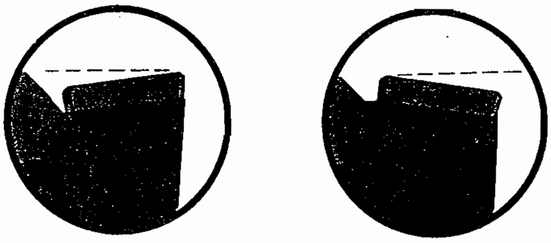


Fig. 3.27. Shape of the inserts.

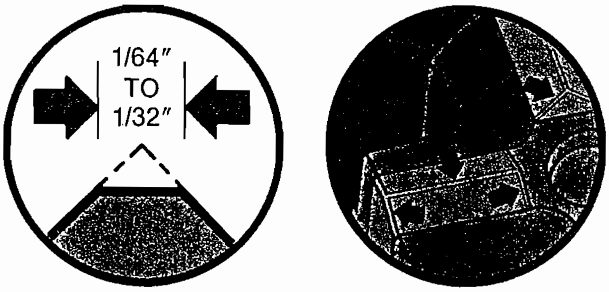


Fig. 3.28. Chamfering of edges.

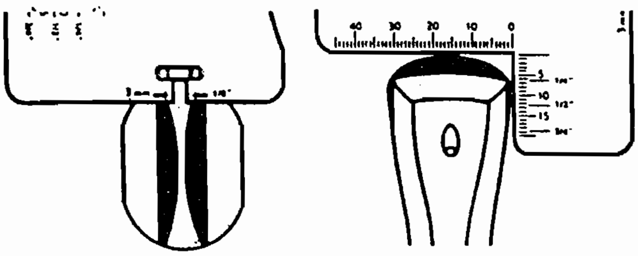


Fig. 3.29. Wear control.

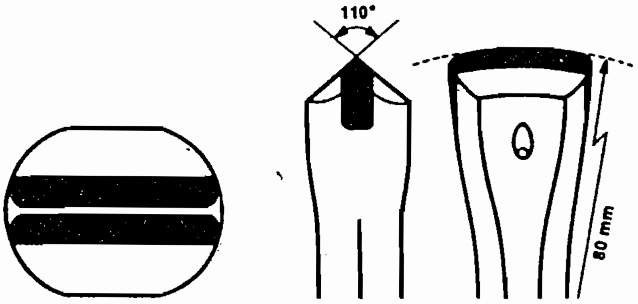


Fig. 3.30. Grinding of integral drills.

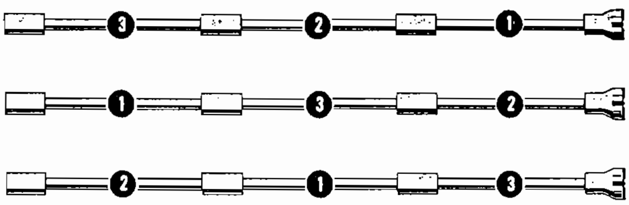


Fig. 3.31. Extension rod rotation.

Fig. 3.32. Rod storage.

Table 3.8.

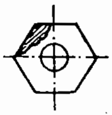
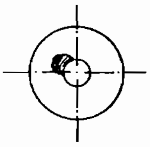
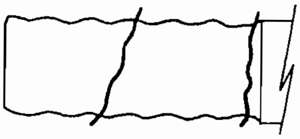
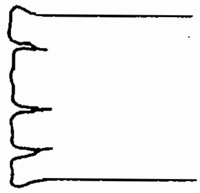
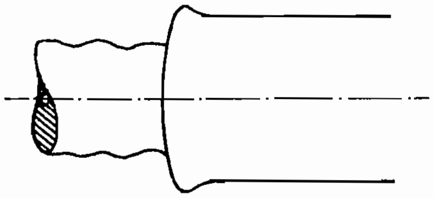
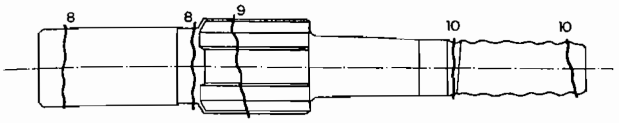
Problem	Probable cause
1. Outer surface rod damage falls, or surface defects. 	1. Poor handling with blows in steel.
2. Corrosion in the bore accelerated by high stress conditions. 	2. Inner defect in steel e.g. an oxidized inclusion.
3. Rod breakage at the end of the threads with couplings. 	3. Threads worn out in rods and couplings, and oscillating enter in the movement of drill steel. Migrating couplings and poor machining on drill steel.
4. Blockage in flushing channels and jammings of the drill steel.	4. Insufficient air flow and excessive feed in cracked, muddy ground.
5. Damage to the end of the coupling which expands and exhibits jagged cracks. 	5. Drilling rod firmly screwed into coupling or off line drilling. Poor heat treatment of steel.
6. Couplings that are broken or split open.	6. Slippage between hammer and centralizer, oscillation of the rods during drilling or incorrect rod threads.
7. Coupling cannot be removed from drill rod. 	7. The coupling hits against the centralizer.

Table 3.8. Cont.

8. Breakage of the shank adaptor or shank.	8. Worn chuck driver bushing, overfeed, lack of lubrication or broken piston.



9. Shank adaptor breakage through the splines.	9. Worn chuck driver bushing, excessive rotation load, lack of lubrication, or broken piston.
10. Shank adaptor breakage at or near the threads.	10. Damaged drill rods or chipped at the ends, damaged couplings or off line drilling. Poor machining or heat treatment of the steel.
11. Broken buttons or sheared off the steel body of the bit.	11. Excessive rotation of the bit. Excessive feed in hard rock. Drilling of cracked stone formations or poorly sharpened bits.
12. Complete loss or tearing off of button.	12. Insufficient feed and lack of contact between bit and rock.
13. Excessive wear flat of gauge buttons.	13. Excessive rotation, overfeed, and rock too abrasive.

3.9 CARE AND MAINTENANCE OF DRILL STEEL

The recommendations that should be followed when using drill steel are:

1. Invert the ends of the extension rods so as to distribute the thread wear.

2. Rotate the rods on the drill strings so that they all have the same meterage, Fig. 3.31.

3. Protect the extension rods against corrosion and dust, storing them in a proper manner and handling them with care, Fig. 3.32.

4. Oil the threads and coupling sleeves with each use.

5. Tighten the couplings to the stop during operation so as to improve energy transmission and avoid overheating of the steel.

6. Use the proper tools for loosening the couplings.

7. Do not reuse rods and coupling that have excessive thread wear.

3.10 GUIDE FOR IDENTIFYING ACCESSORY FAILURE AND ITS CAUSES

Table 3.8 shows the different types of breakage for drilling accessories, such as rods, threads, coupling sleeves, shank adaptors and bits, and their probable causes.

Any damage or flaw should be analyzed and identified to be able to correct the originating source or operative practice.

REFERENCES

Anonymous: *Rock Drilling Seminar.* Mining Magazine, July, 1979.
Atlas Copco: *Manual Atlas Copco.* 1984.
Fagersta-Secoroc: *Accesorios de Perforación.* 1974.
Gardner Denver: *Rock Drilling Data.*
Ingersoll-Rand: *La Boca de Botones Contra la Roca.*
Kometa Oy.: *Accesorios de Perforación.* 1986.
Oliver, J.: *Factors Influencing the Selection & Use of DTH Button Drilling Applications.* II Symposium Nacional de Selección de Maquinaria en Minería e Industrias de la Construcción, 1990.
Sandvik, AB.: *Rock Drilling Manual – Drill Steel Applications.* 1979.
Sandvik-Coromant: *Manual de Perforación de Rocas – Teoría y Técnica.* 1983.
Tamrock: *Handbook of Underground Drilling.* 1983.
Tamrock: *Handbook of Surface Mining.* 1989.
Tantarimaki, K.: *Top-Hammer.* World Mining Equipment. September, 1990.
Timken: *Brocas de Percusión para Roca.* 1981.
TRW, Inc.: *Percussion Drilling Equipment Operation and Maintenance Manual.* 1985.

CHAPTER 4

Rotary drilling with rolling tricone bits

4.1 INTRODUCTION

Until 1949, almost all blastholes were drilled with rotary percussive rigs and only in very soft rocks were rotary drills applicable with drag bits.

The opening of large surface coal mining operations in the United States, with overburdens that went up to 40 m, and the appearance on the market of a low cost bulk explosive of high energy efficiency, such as ANFO, were events that stimulated the manufacturers of rock drills to design large capacity equipment that were capable of giving high penetration rates

At the same time roller tricone bits became popular in mining, developed in the oil fields from around 1907, and compressed air came into use for flushing away the drill cuttings.

The blasthole diameters varied between 2″ and $17^{1/2}$″ (50 to 444 mm), with the most frequent range for surface mining from 6 to $12^{1/4}$″ (152 to 311 mm). Larger diameters are limited to mines with large productions, and under 6″ is almost never used because of the limited wear of the tricone bits due to the reduced size of the bearings.

This drilling method is very versatil, as it covers a wide range of rocks, from the very soft where its application started, to the very hard where its use has replaced other once popular systems, such as Jet Piercing in taconites.

As rotary drilling with roller tricone bits is the most widely used method, this chapter will focus its attention on the large rigs that are capable of giving high feed force on the bit, considering that the units that work with drag bits are of a more simple design and smaller in size.

Rotary rock drills essentially consist of a power source, a string of pipes, individual or connected in series that transmit the load, rotation and flushing air to a bit with steel teeth or tungsten carbide inserts that, in turn, acts upon the rock, Fig. 4.1.

4.2 MOUNTING AND PROPULSION SYSTEMS

There are two basic mounting systems for rotary drilling rigs: on crawler tracks or on rubber tyres. The factors that dictate which type of mounting should be chosen are the ground conditions and the degree of movability required.

If the work surface has steep grades, is uneven or offers maneuvering difficulties, crawler mounting is the most indicated as it offers maximum stability, maneuverability and flotability.

A heavy axle at the drill end of the machine and a pivot axle permit the rig to oscillate and maintain full crawler ground contact at all times, Fig. 4.2.

The majority of the large rigs are mounted on rugged crawlers, as these can support heavier loads with less pressure on the ground when moving from one place to another.

The drills that are mounted on tractor type crawlers are useful in difficult and abrupt terrain such as can be found in civil engineering.

The main disadvantage of crawler mountings is its low translation velocity, 2 to 3 km/h. Therefore, if many benches are to be drilled and they are quite far away from each other, it would be more advisable to mount the rig on a truck, which moves an average of ten times faster. However, in large operations the equipment is not usually moved very much as they drill a high number of blastholes in a reduced area.

Much lighter rigs are usually mounted on trucks, with chasis of 2 or 3 axles, and only those of larger dimensions, more than 60.000 pounds of feed force, are mounted on 4 axle chasis. During drilling, these units are supported by 3 or 4 hydraulic jacks which also serve to level the machine.

4.3 POWER SOURCES

The primary sources of power can be: diesel or electric engines.

In rock drills with a drilling diameter above 9″ (230 mm), electric energy is normally used, at half tension, supplying the drill with alternating current (AC) by means of a four conduct power cable covered with rubber.

The small and medium sized rigs, which are usually mounted on trucks, can be run by one or two diesel engines.

The average distribution of the power installed on these rigs for different operations and mechanisms is as follows:

- Elevation and translation movement: 18%
- Rotation: 18%
- Thrust load: 3%
- Levelling: 2%
- Dust collector: 3%
- Flushing and removal of drill cuttings with compressed air: 53%

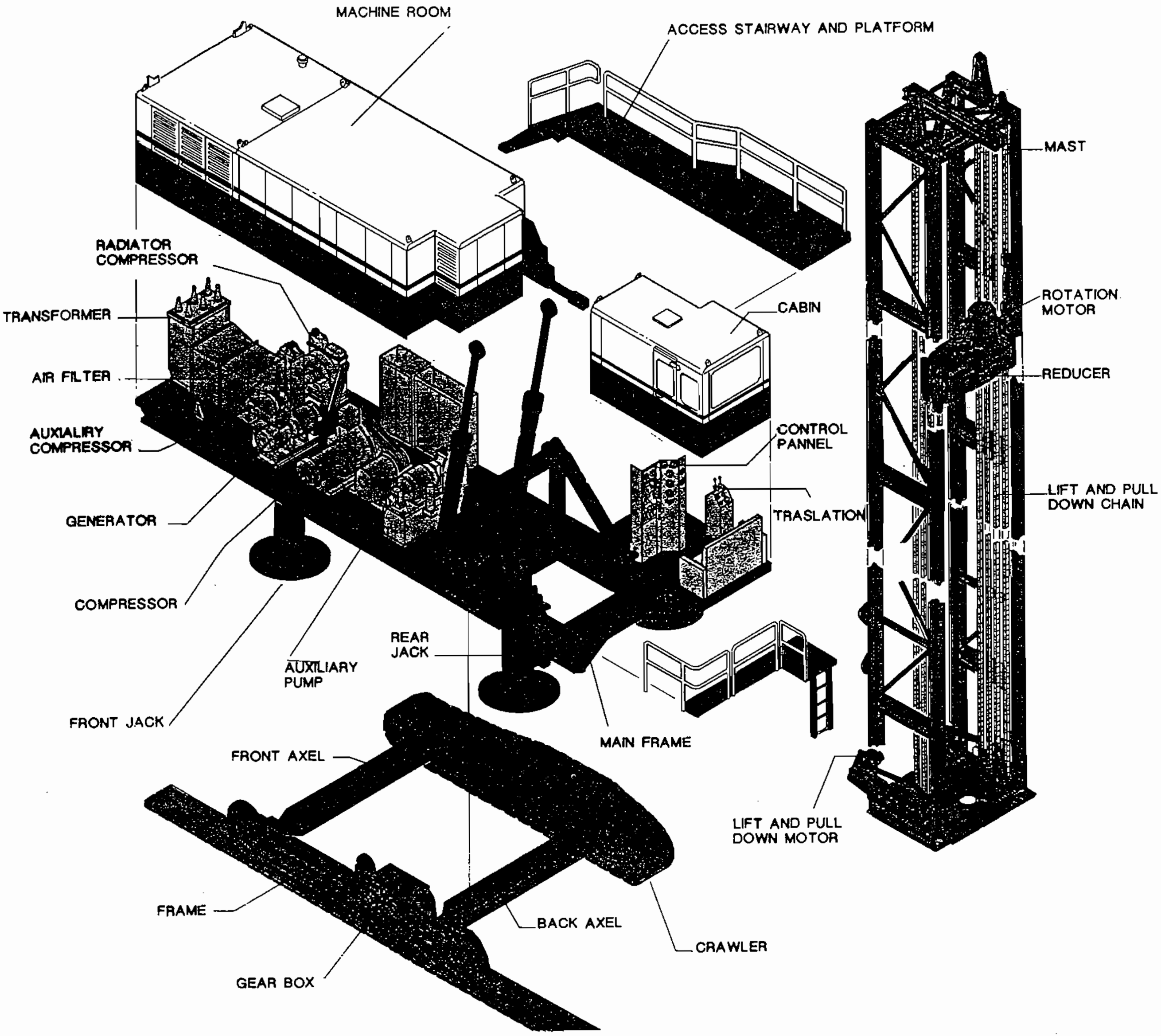


Fig. 4.1. Principal components of an electrically driven rotary rig (Marion).

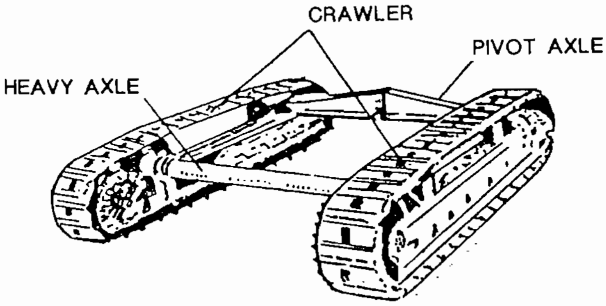


Fig. 4.2. Rugged crawler mounting.

– Auxiliary equipment: 3%

When diesel power is used, this can be from the truck engine, Fig. 4.3, or with an independant engine. Nowadays, the latter is usually the case owing to the different characteristics required.

There are also diesel-electric drills that have been designed for large production mines without an electric power layout.

The electric rigs have lower maintenance costs, 10 to 15%, than those driven by diesel power. The latter are chosen when there is no reliable electric power source near the operation, or when the rig is mounted on a truck.

4.4 ROTATION SYSTEMS

With the purpose of making the drill pipes rotate and the transmission of the rotary torque, the drilling rigs have a rotation system mounted on a frame that slides up and down the mast.

The system of top head drive can be run by an electric or hydraulic engine. The first is the most commonly used for large rigs, as the continuous current engines are easily regulated, from 0 to 150 r.p.m. In the older designs, the Ward Leonard system was used, and in the more modern the thyristor converter D.C. motor system.

The hydraulic system consists of a closed circuit with a constant pressure pump and a rotary torque converter

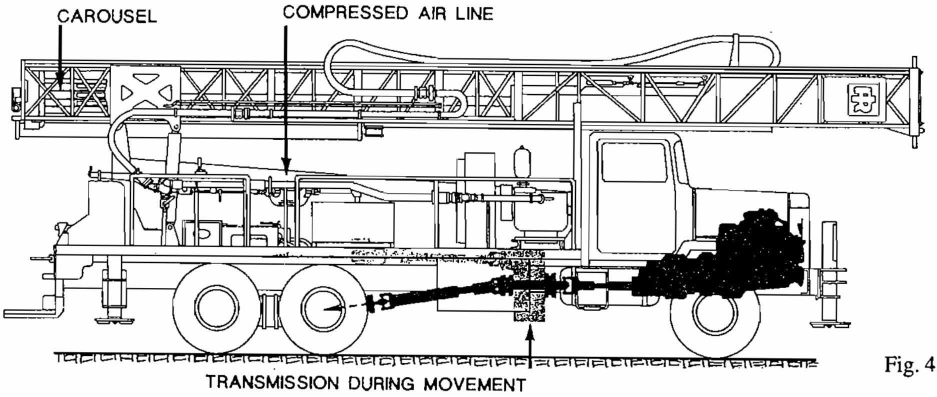


Fig. 4.3. Truck engine diesel drive for a rock drill.

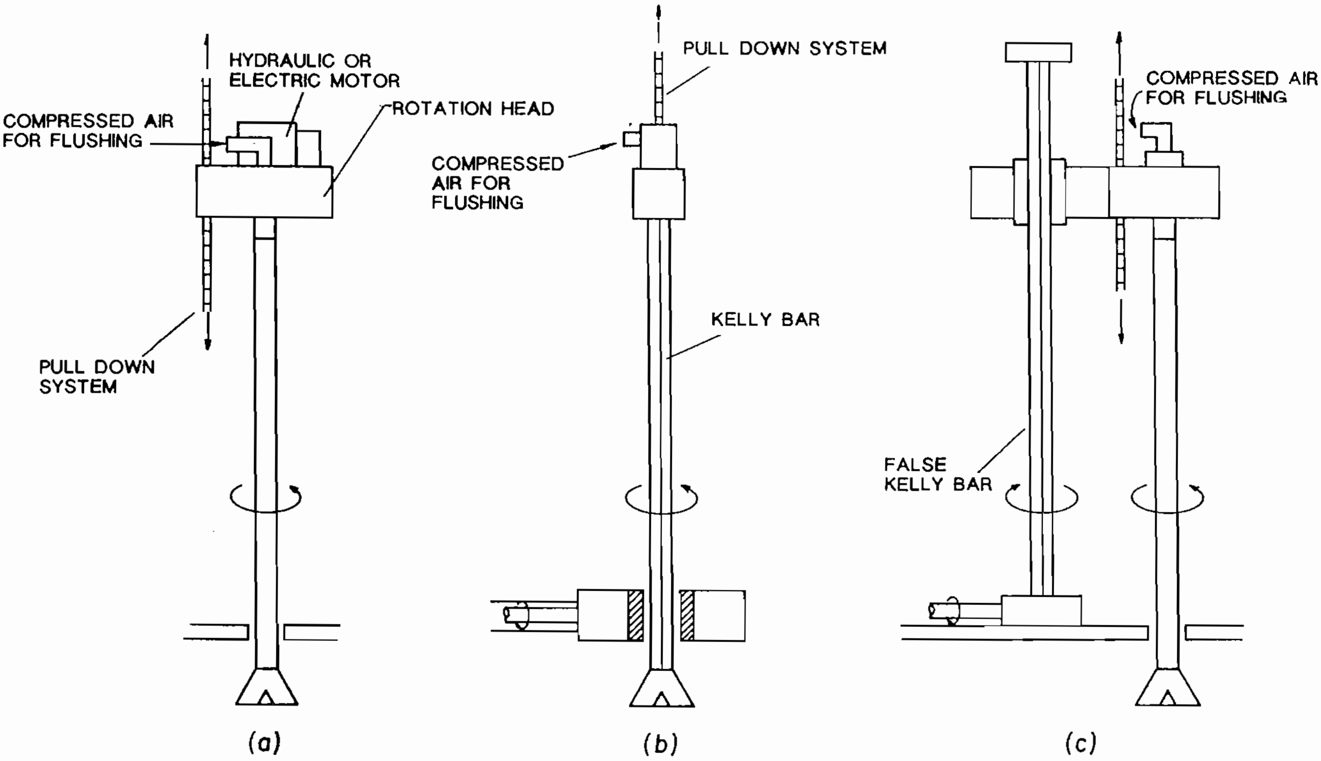

HYDRAULIC OR ELECTRIC MOTOR
COMPRESSED AIR FOR FLUSHING
ROTATION HEAD
PULL DOWN SYSTEM
(a)
PULL DOWN SYSTEM
COMPRESSED AIR FOR FLUSHING
KELLY BAR
(b)
COMPRESSED AIR FOR FLUSHING
FALSE KELLY BAR
(c)

Fig. 4.4. Rotary drive options: (a) top head drive, (b) Kelly bar drive and (c) false Kelly bar drive.

with which the rotary speed of the top head drive hydraulic engine can be varied. This method is widely used on small and medium rigs.

The mechanical or indirect systems are the Kelly bar, or rotary table, which is very popular in the oil fields but not much used in mining machinery, and the so-called false Kelly bar, which are shown in Fig. 4.4.

4.5 PULLDOWN/HOISTING SYSTEMS

In order to achieve a good drilling rate in rock, a certain pulldown or feed is necessary. This depends upon the compressive strength of the rock and the blasthole diameter. As the weight of the bars is not sufficient to obtain the precise thrust load, extra forces must be applied through hydraulic energy tranmission.

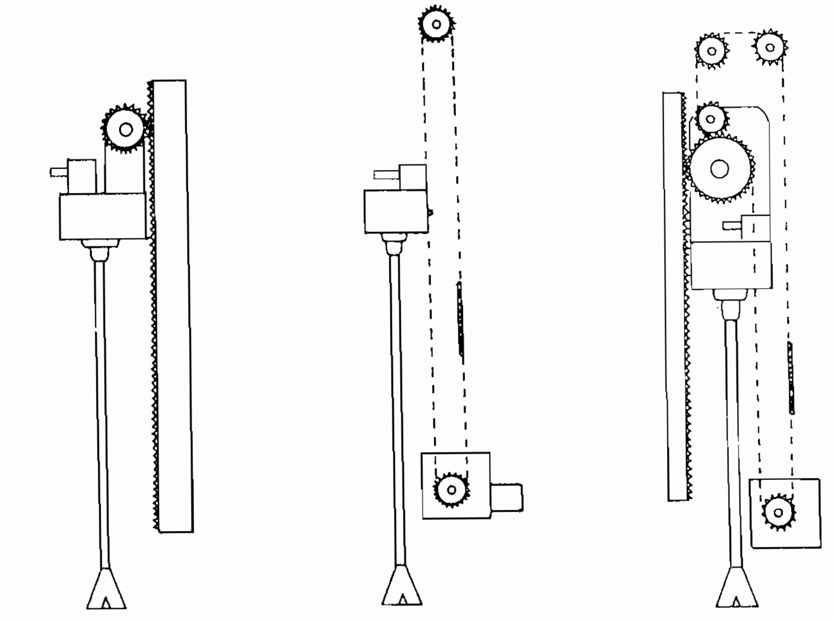
There are four systems. The first three, shown in Fig. 4.5, are known as a) direct rack and pinion, b) direct chain, and c) rack and pinion with chain.

The fourth system, Fig. 4.6, consists of one or two hydraulically driven cylinders. It has the following advantages: light weight, absorbs impacts, indicates wear or fatigue level and is easy to change or adjust.

These feed mechanisms allow, apart from supplying a perfectly controlled, balanced feed force, the hoisting of the pipes that constitute the drill string.

The weight of the whole set up behaves as a reaction against the feed applied to the bit, from which one can deduce that the weight of the rock drill should be above and normally double that of the maximum intended thrust.

The hoisting rate of the string is usually from 18 to 21 meters/min; higher rates are not recommendable due to vibration problems.



a) DIRECT RACK AND PINION b) DIRECT CHAIN c) RACK AND PINION WITH CHAIN

Fig. 4.5. Pulldown or feed options.

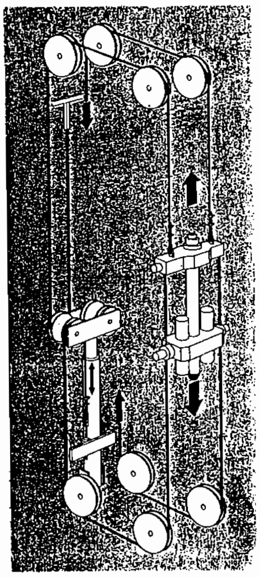


Fig. 4.6. Pull down system with cylinder (Ingersoll-Rand).

4.6 MAST AND PIPE CHANGER

The masts that accomodate the drill steel and rotary head are the most critical structure on the drill and should be designed to resist bending from weight, the thrust load and the strains originated by the rotary torque. The most common designs are constructed in a truss pattern of tubular or normal section steel. The modern rigs have a structure made of box type girders that can withstand longer single-pass masts and high rotary torques.

Masts are usually hinged and connected to hydraulic cylinders that permit them to be lowered to a horizontal position for moving. Mast elevation time is usually around 2 to 5 minutes.

Inclined hole drilling is usually harmful because of the fatigue forces placed upon the mast and drill pipes, apart from a lower pull down load and flushing difficulties, all of which lower production which, in hard rocks, can

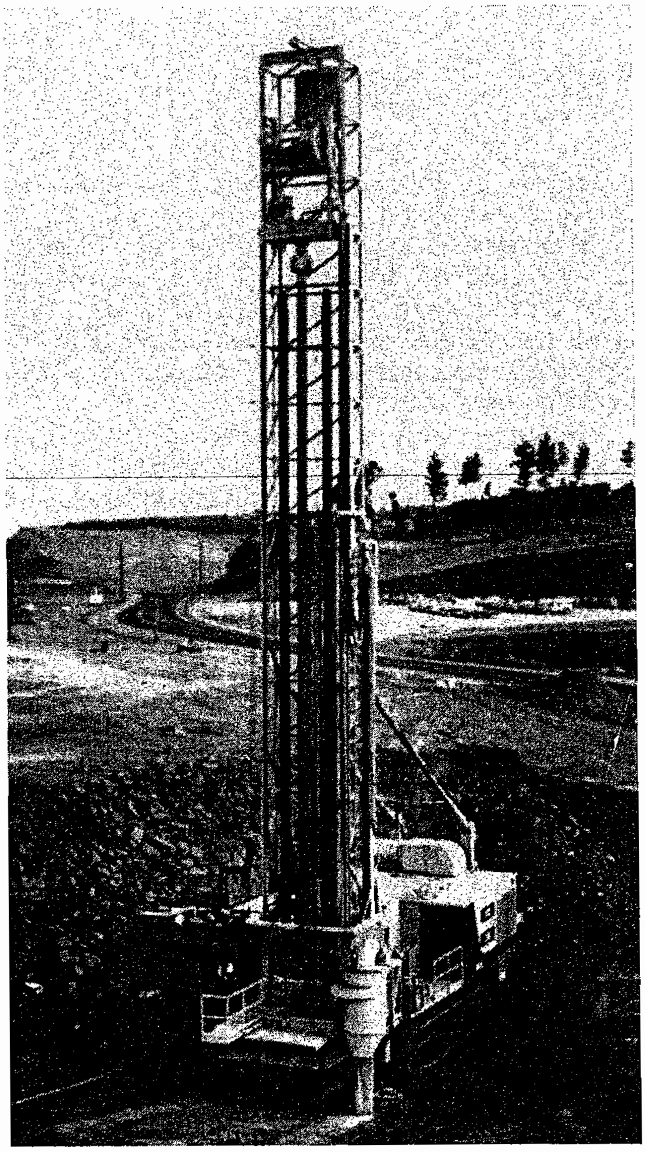


Photo 4.1. Crawler mounted rotary rig, 49 R. (Courtesy of Bucyrus-Eire)

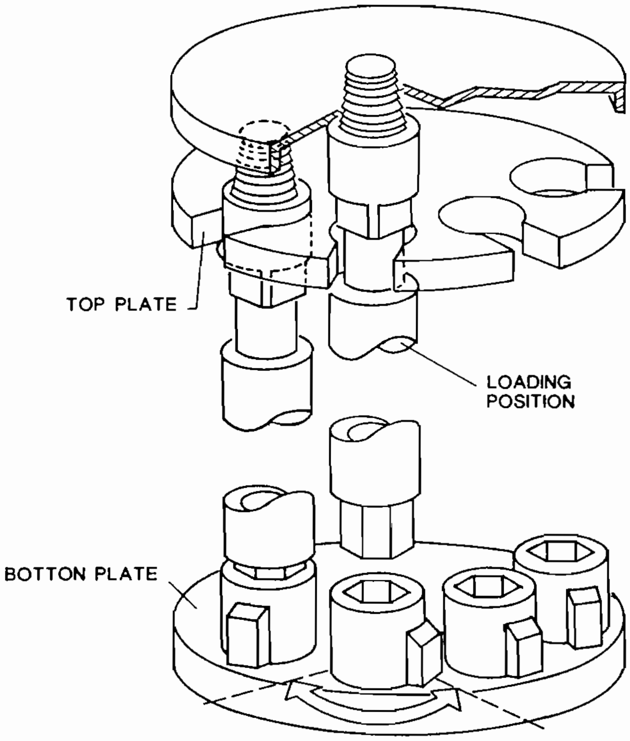


Fig. 4.7. Carouselle type pipe changer.

reach 20%. The pinning or inclination can be as much as 30% in 5° intervals.

Even though it is recommended that the machine selected be capable of drilling the blastholes with only one pipe, there is always the possibility of having to drill longer holes. Therefore, the mast should carry a carouselle type pipe changer as well as a mechanism for placing and removing them.

The rigs are equipped with tray like sistems with from one to three pipes, or carouselle type with four or more pipes with a drilling capacity of 50 to 60 meters. Both systems are hydraulically driyen, Fig. 4.7.

Pipe changing usually takes between 2 to 6 minutes each.

4.7 CONTROL CABIN

The control cabin, pressurized and air conditioned, has all the controls and instruments required for handling the rig during drilling, as follows:

- Control of the principal motor and gear box.
- Control of elevation and lowering of the mast.
- Control of the leveling jacks.
- Control of pulldown on the bit.
- Control of water injection.
- Control of the carouselle, etc.

Usually the cabin is located near the mast so that all movements of the pipes during work can be observed.

4.8 SYSTEM FOR FLUSHING DRILL CUTTINGS

Compressed air carries out the following functions:

- Cooling and lubrication of the roller tricone bearings.
- Cleaning the bottom of the blasthole.
- Bailing out of the drill cuttings with adequate upward velocity.

The air circulates through a pipe from the compressor to the mast and from it, by a protected flexible hose, to the rotary head from which it enters into the drill pipe and eventually arrives at the bit. The air flushes out from passages in the bit through the teeth to swirl up the cuttings and bail them upwards to the surface. If the tricones are large and the flow of air insufficient, the cuttings fall back to the bottom to swirl around until they reach a large enough size to be swept upwards. Lack of air, therefore, produces an unnecessary waste of energy, a lower penetration rate and more wear on the bit. On the other hand, if the upward velocity is too high, the centralizer and the drill pipes will suffer more wear.

If the rock density and the diameter of the chips are known, two equations can be used to calculate the minimum upward velocity:

$$V_a = 573 \times \frac{\rho_r}{\rho_r + 1} \times d_p^{0.6}$$

and

$$V_a = 250 \times \rho_r^{1/2} \times d_p^{1/2}$$

where: V_a = Minimum upward velocity (m/min), ρ_r = Rock Density (g/cm^3), d_p = Chip diameter (mm).

The air flow necessary is determined by:

$$Q_a = A_b \times V_a = V_a \times \frac{(D^2 - d^2)}{1.27}$$

where: A_b = Cross-sectional area of the annulus between the blasthole wall and the drill pipe (m^2), Q_a = Required air flow (m^3/min), V_a = Upward velocity (m/min), D = Blasthole diameter (m), d = Drill pipe diameter (m).

Another equation to determine the approximate flow is:

$$Q_a = 224 \times D^{3/2}$$

where: Q_a = Air flow (m^3/min), D = Blasthole diameter (m).

The recommended bailing velocities, in function with the type of rock, are the following:

Table 4.1.

Type of rock	Minimum velocity		Maximum velocity	
	(m/min)	(feet/min)	(m/min)	(feet/min)
Soft	1.200	4.000	1.800	6.000
Medium	1.500	5.000	2.100	7.000
Hard	1.800	6.000	2.400	8.000

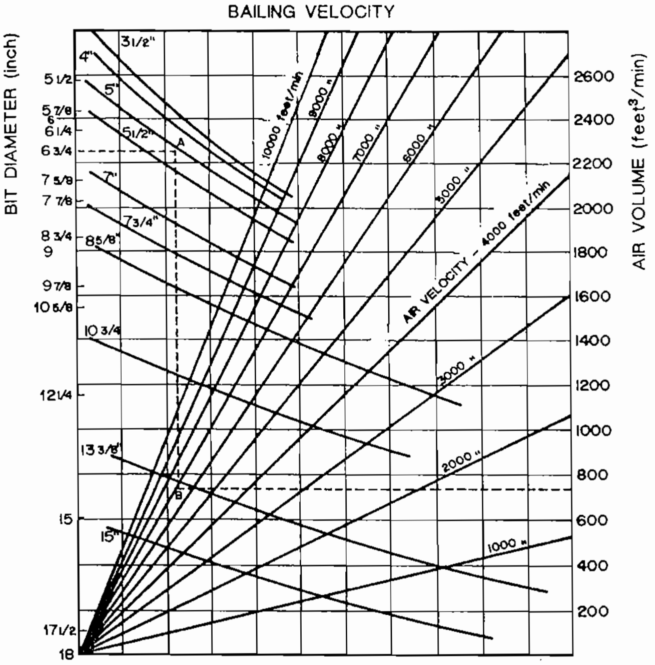


Fig. 4.8. Drill pipe sizing.

Thus, the drill pipe diameter recommended, according to the rock type, should be 3″ (75 mm) smaller than that of the tricone bit in soft material, 2″ (50 mm) in medium rocks and 1½″ (38 mm) in hard rock, as the particle or chip size decreases with an increase in rock compressive strength.

From the nomograph below, the diameter of commercial drill pipes can be determined more exactly, once the air flow, upward velocity and blasthole diameter are known.

When the compressive rock strength is under 100 MPa, the high drilling rate achieved retains the cuttings in the blasthole if the annulus or ring of space is not large enough. The following should comply:

Blasthole area divided by the annulus area = 2

which is the equivalent of:

Drill pipe diameter divided by blasthole diameter = 0.7

Normally low pressure compressors are used in rotary drills, 50 p.s.i. (350 kPa). However, the number of rigs that have medium to high pressure compressors, 100-150 p.s.i. (700-1050 kPa), is increasing owing to improved cooling of the bearings and the possibility of using down-the-hole hammers.

4.9 DRILL STRING

The drill string, Fig. 4.9 is composed of the rotary coupling, the drill pipe, the stabilizer and the rolling cone or rotary tricone rock bit.

4.9.1 *Rotary coupling*

This element transmits the rotary torque from the rotary head to the drill string which is underneath.

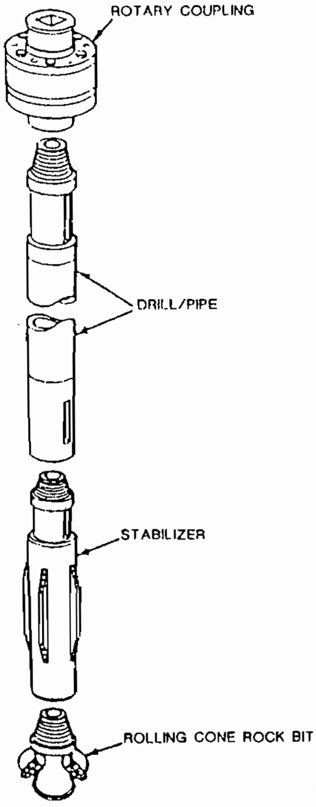


Fig. 4.9. Drill string.

4.9.2 *Drill pipe*

The length of the drill pipes depends upon the length of the blasthole. They transmit the pulldown or feed force to the bit and to canalize the compressed air required to clean the hole and cool the bearings. The pipes are usually made of steel with a 1″ thickness (25 mm) and sometimes up to 1½″ (38 mm). The most common threads used in the couplings are API, BECO, or similar.

4.9.3 *Stabilizer*

The stabilizer is placed on top of the drill bit, Fig. 4.10, and its mission is to make the tricone bit rotate correctly depending upon the axis of the blasthole and avoid oscillations and bending of the drill steel.

The advantages of its use are:

- Fewer blasthole deviations, above all in inclined drilling.
- Less wear on the tricone and better drilling rates due to improved feed force.
- Less skirt, gauge insert and bearing wear.
- Improved blasthole wall stability as the drill pipes do not bend.
- Improved bulk explosive loading, as the distribution is more uniform.

The stabilizer should have a diameter similar to that of the hole, usually ⅛″ (3 mm) smaller than that of the tricone bit.

There are two types of stabilizers: blade type and roller type.

The blade type is least expensive but the disadvantages are that it must be used with wear bars, it reduces the available rotary torque and is unstable in hard ground after drilling the first holes.

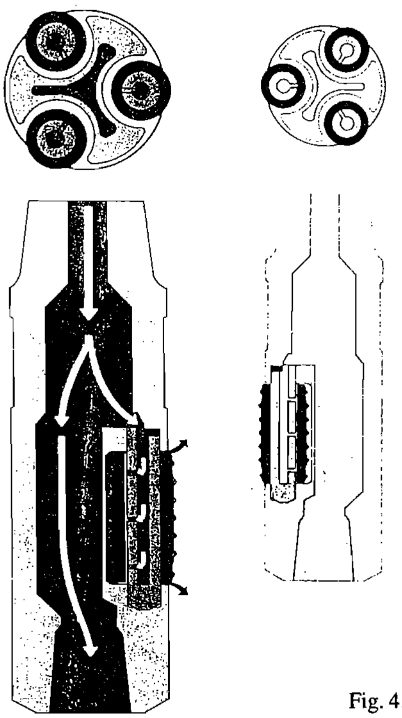


Fig. 4.10. Roller type stabilizer.

The roller type stabilizers with tungsten carbide buttons do not require much rotary torque, are more expensive and also more efficient than the blade type.

4.9.4 *Single pass drilling*

The use of high masts, up to 27 m, which allow each hole to be drilled in a single pass with lengthening the drill string has the following advantages:

- No added drill pipes which takes from 2 to 6 minutes each.
- Reduced thread wear.
- 10 to 15% increase in production.
- Easier blasthole cleaning.
- Permits a continuous air flow through the bit which is especially interesting in blastholes with water.
- Reduces loss of feed and rotation force transmission as there are no unions between pipes.

There are a few disadvantages to single pass drilling such as:

- Higher masts are more unstable, especially in the rotary head.
- The mast requires stronger anchoring to the rig.
- More care has to be taken when moving.
- The feed transmission chain needs an improved design.

4.9.5 *Vibration and shock subs*

Since 1967 a series of vibration and shock absorption systems have been developed which give the following benefits:

- Lower rock drill maintenance cost by reducing axial and torsional shock loads transmitted to the mast.
- Increase in drilling rate by improving bit contact with the rock and enabling a better use of the combination feed force/drilling rate for each type of formation.

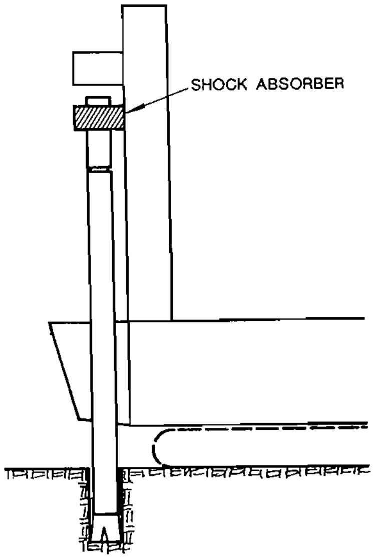


Fig. 4.11. Anti-shock coupling placement.

- Increased service life of the bit as the impacts to the bearings and cutting structure are absorbed.
- Less noise level in the operator cabin as the direct contact rotary head/drill bar is eliminated.

Anti-shock couplings should be used in broken ground, alternating hard and soft layers and in hard formations.

The type of shock subs used are:

- Horizontal shock absorbers.
- Vertical shock absorbers (swivel mounted).
- Nitrogen shock absorbers (gas spring)

Horizontal shock absorber

Performs as a flexible and compressible unit and reduces vertical and transversal vibration. Field tests have shown an increase in penetration rate of 5% in soft rocks and 20% in hard rocks, with a 25% increase in tricone life. One of the characteristics of this system is that it only has two elements that can be worn out.

Vertical shock absorber (swivel mounted)

This type has an assembly of 18 elastic segments that are mounted vertically, giving a performance similar to that of the horizontal shock absorber, Fig. 4.12.

Nitrogen shock absorber (gas spring)

This system uses pressurized nitrogen. Its main disadvantage is the high initial cost and maintenance.

4.9.6 *Blasthole reamers*

This is an interesting practice as it permits the use of explosive columns similar to spherical charges. The advantages of the system of *chambering* against the conventional uniform blasthole can be summed up as follows:

- Less volume of rock drilled.
- Higher production output.
- Less manuevering time.
- Lower volume of stemming, and
- Improved muckpile profile for front end loader.

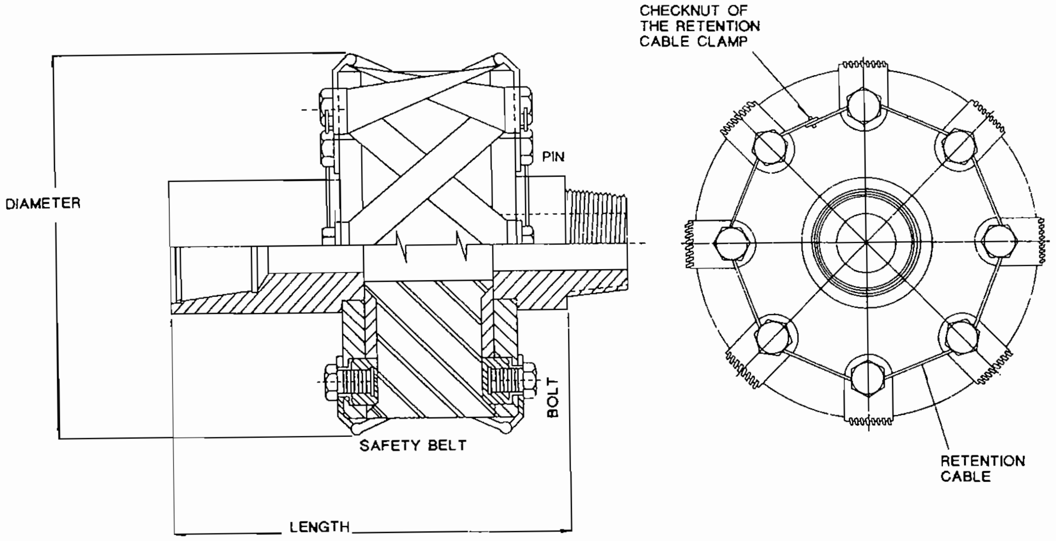


Fig. 4.12. Vertical shock absorber (B.J. Hughes, Inc.)

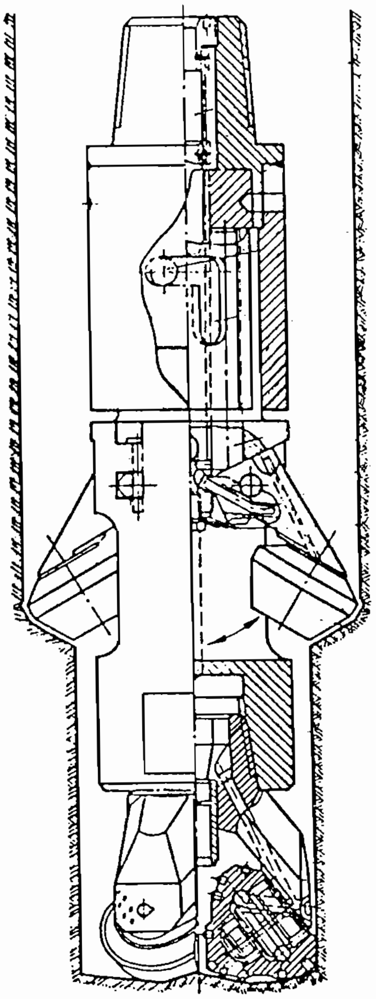


Fig. 4.13. System for widening blastholes.

4.10 AUXILIARY ELEMENTS

4.10.1 *Dust elimination*

A large amount of dust builds up during drilling and if not eliminated, apart from affecting the health of personnel, it can create maintenance problems in the rock drill. Dust suppression can be carried in two ways:

- Wet system.
- Dry system.

The wet system consists in adding a small amount of water, with or without foam, into the flushing air. The dust that has formed in the bottom of the hole becomes caked and is bailed upwards with the cuttings.

This system is simple but has disadvantages, which are:

- Lower service life of the tricone bit – between 15 to 20%.
- If too much water is injected the dust can become pasty, thick and abrasive, which is difficult to bail out and could cause great wear on the drill string.
- In cold climates it produces operative problems.

The dry dust suppression system consists of a dust collector mounted on the rig and composed of cyclones and filters. The big advantage of this method is that it does not affect the bits and is very efficient. However, when water is present in the hole its efficiency is quite reduced and the maintenance is high.

The operators cabin and the engine room are usually pressurized to keep the dust out.

4.10.2 *Leveling*

When the machine is drilling, its weight rests on leveling jacks which are pinned to the drills main structure and can be regulated from the operator cabin. Each rig usually has from three to four jacks and it takes about 1 minute to set them up.

A hydraulic jack at each corner of the machine is usually the best arrangement to evenly distribute the weight, reducing strain on the rig, mast vibrations and general damage, Fig. 4.14.

4.10.3 *Stability*

In order to obtain high productivity, the rigs should be able to move with the mast and drill string in the vertical

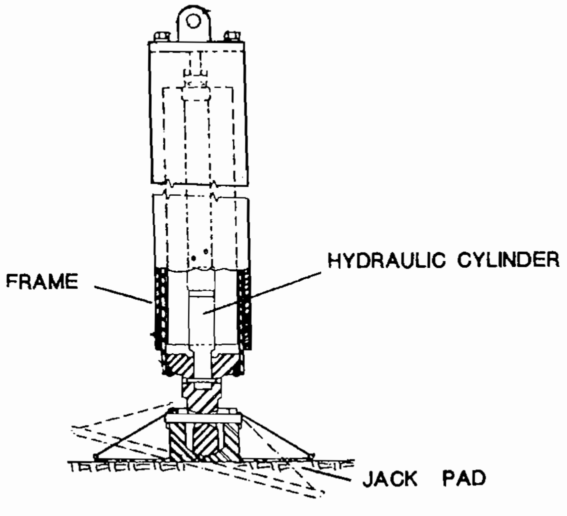


Fig. 4.14. Hydraulic jack.

position. For this reason, the equipment should be designed in such a manner that its center of gravity is as low as possible and close to the centerline of the crawler. The crawlers can be oversized to increase stability and carry ballast to keep the rig in balance.

4.10.4 *Gradeability*

Rigs on crawlers are capable of propelling up 10 to 12% grades for one hour and reach peak gradeability of 20% for a short time.

4.10.5 *Oil or grease injection*

The injection of oil into the flushing air provides extra lubrication to the bearings of the tricone bit, helping to cut down on wear, Fig.4.15.

If the flow is excessive, the air passages in the bearings can become blocked and premature failure could occur, as well as the dust taking on a pasty consistency which is an obstacle for uphole bailing.

If compressors with sliding-vanes are used, it has been found that the service life of the bits increases noticeably, as the air carries a small amount of oil with it. For this reason, if the compressors that are mounted on the rigs are of rotary screw type then it is recommended that oil be injected into the flushing air.

4.11 OPERATIVE PRACTICE. DRILLING PARAMETERS

The internal parameters that intervene in rotary drilling are:

- Feed or pulldown force on the bit.
- Rotation speed.
- Bit wear.
- Blasthole diameter, and
- Flushing air flow.

The external parameters are the following:

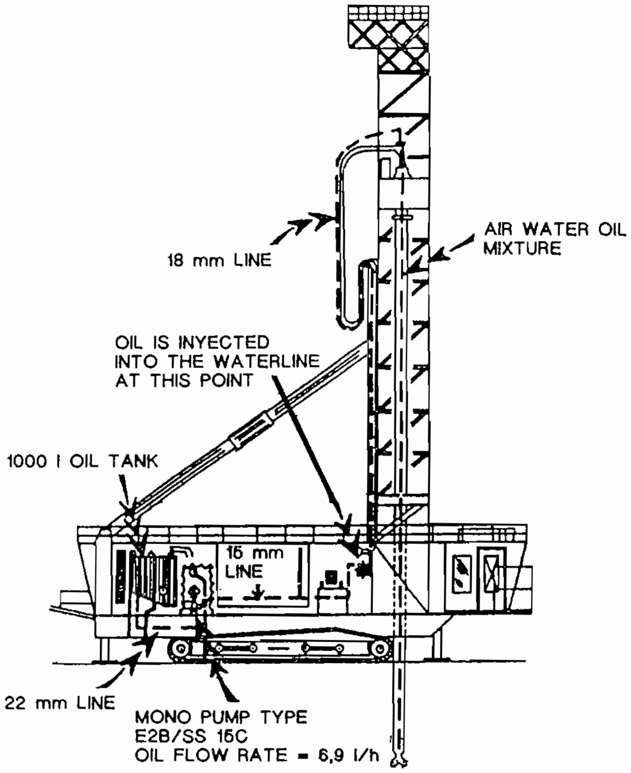


Fig. 4.15. Oil injection system in a rotary rock drill.

- Characteristics and compressive strength of the rocks, and
- Operator efficiency.

4.11.1 *Feed or thrust load on the bit*

The feed force applied to the bit should be sufficient to overcome the compressive strength of the rock, but not excessive to avoid premature or abnormal failure of the tricone bit.

The drilling rate increases proportionately with the feed, up to the point where the bit becomes locked against the rock as the teeth or inserts bury into the material, Fig. 4.16, or because of the high drilling rate and the large volume of cuttings produced the blasthole is not adequately cleaned.

In hard formations, a high thrust force on the bit can produce breakage of the inserts before the bit becomes stuck or the hole cleaning is defective. Also the bearing life is diminished, but not necessarily the length drilled by the rolling cone bit.

When rock is drilled, the tricones can work in three different situations, Fig. 4.17.

a) With insufficient feed or pulldown.
b) With efficient advance, and
c) When the tool becomes buried.

The *minimum pulldown* under which a rock is not drilled, can be estimated from the following equation:

$$E_m = 28.5 \times RC \times D$$

where: E_m = Minimum pulldown (pounds), RC = Compressive strength of the rock (MPa), D = Diameter of the tricone bit (inches).

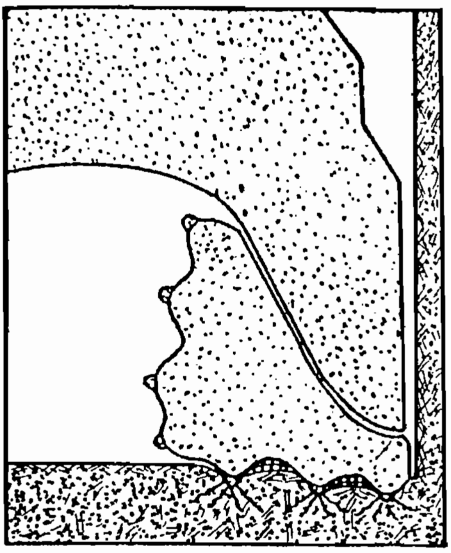


Fig. 4.16. Rock breakage. Excessive feed.

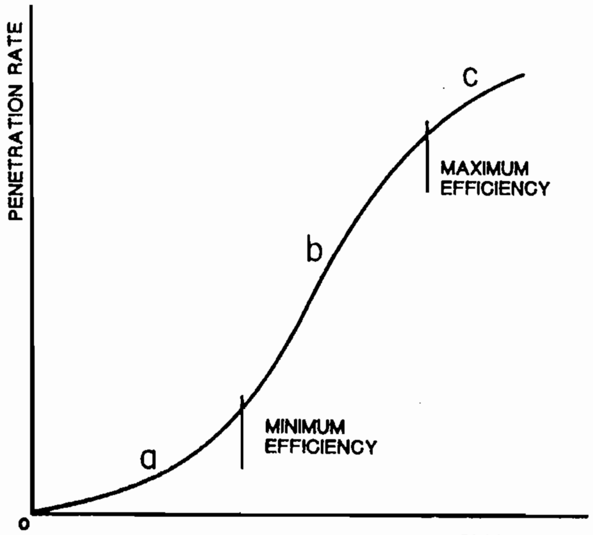


Fig. 4.17. Pulldown effect on drilling rate.

Table 4.2.

Diameter of the bit (inches)	Limit pulldown (lbs)
$5\frac{7}{8}$	21.000
$6\frac{1}{4}$	31.000
$6\frac{3}{4}$	37.000
$7\frac{7}{8}$	50.000
9	65.000
$9\frac{7}{8}$	79.000
$12\frac{1}{4}$	121.000

The *maximum pulldown*, above which the tricone is buried, is considered to be double the previous value.

$$E_m = 2 \times E_m$$

limit pulldown that a roller tricone can withstand is in function with the size of its bearings which, at the same time, depends upon the diameter of the bit:

$$E_L = 810 \times D^2$$

where: E_L = Limit pulldown on the tricone (pounds), D = Diameter (inches).

Table 4.2 gives the limit values for tricones of different diameters.

4.11.2 *Rotation speed*

Penetration rate increases with rotation speed in a proportion that is slightly lower than the unit, up to a limit imposed by the bailing out of drill cuttings, Fig. 4.18.

Rotary speeds vary from 60 to 120 r/min for steel toothed roller tricones, and from 50 to 80 r/min for those with tungsten carbide inserts.

Table 4.3 gives the rotary speeds that are adequate for different types of rock.

The rotary speed limit is set by bearing wear, which itself depends upon the pulldown, the cleaning of the blasthole and the temperature; and upon breakage of the inserts produced by impacts against the rock, in which the intensity is proportional to the square of the rotary speed.

4.11.3 *Wear of the bit*

When steel tooth roller tricones are used, the drilling rate goes down considerably as the wear on the bit increases.

Fig. 4.19, shows how, for a half used roller tricone bit, the penetration rate can be reduced by 50 to 75% when compared to that obtained with a new tricone.

4.11.4 *Drilling diameter*

Fig. 4.20 shows how the drilling rate obtained with constant feed force and rotation speed is proportional to the inverse of the squared drilling diameter.

4.11.5 *Air flow*

When drilling is carried out with less air than necessary to

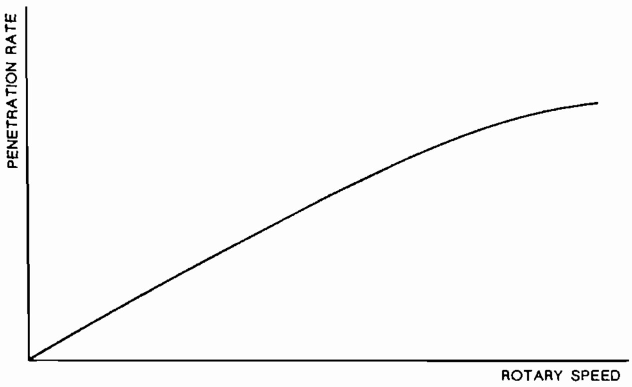


Fig. 4.18. Effect of rotary speed upon the penetration rate.

Table 4.3.

Type of rock	Rotary speed (r/min)
Soft	75-160
Medium	60-80
Hard	35-70

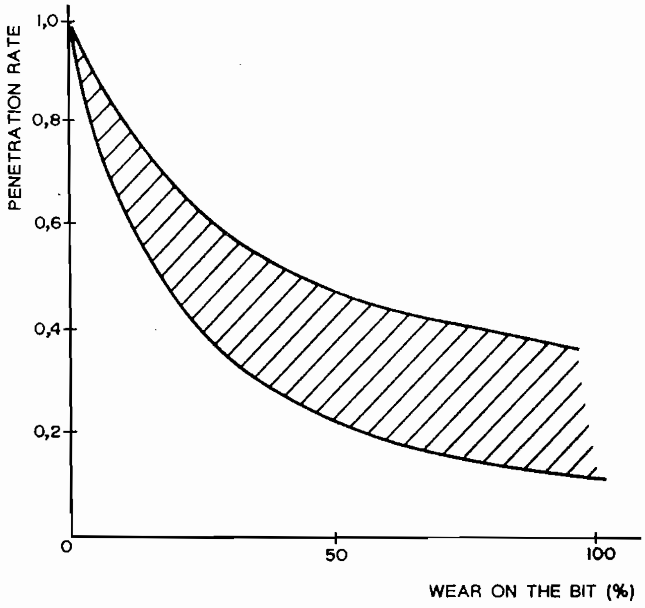


Fig. 4.19. Effect of bit wear on the penetration rate.

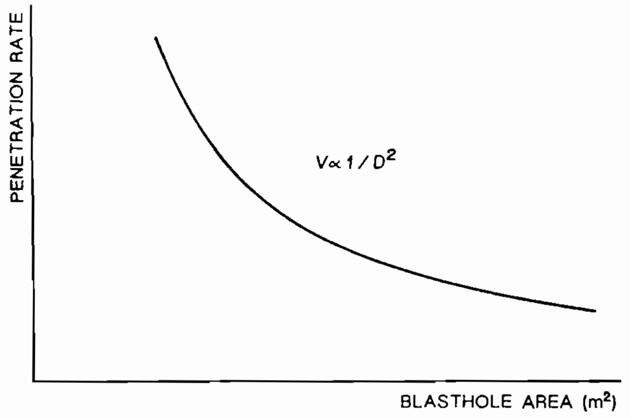


Fig. 4.20. Influence of diameter on the drilling rate.

clean the blasthole, the following negative effects appear:

- Decrease in drilling rate.
- Higher pulldown required to continue drilling.
- Increase in drill damage owing to higher rotary torque necessary to rotate the tricone.
- More wear on the stabilizer, drill pipe and bit.

4.11.6 *Criteria for rock drill selection*

Once the drilling diameter has been determined, which depends upon:

- Required output.
- Capacity and number of loading and haulage equipment.
- Bench height.
- Environmental limitations and
- Operation costs.

And taking into consideration the geomechanics of the rock to be drilled, the following can be determined:

- Rock drill characteristics.
- Type of roller tricone bit.
- Drill steel and accessories.

The adequate design of a rock drill requires taking into account the rotary power necessary to rotate the bit and the method of bailing out the drill cuttings.

Rotary power

The required rotary power is equal to the product of the rotary torque necessary to rotate the bit by the rotation speed.

$$HP_r = \frac{N_r \times T_r}{5.250}$$

where: HP_r = Rotary power (HP), N_r = Rotation speed (r/min), T_r = Rotary torque (lb/ft).

The rotary torque increases with the thrust load on the bit and the depth of the blasthole. Normally, the drills are designed with a torque capacity in the range of 10 to 20 lb/ft for each pound of pulldown load.

When the necessary torque is not known, the rotary power can be calculated from the following equation:

$$HP_r = K \times N_r \times D^{2.5} \times E^{1.5}$$

where: HP_r = Rotary power (HP), N_r = Rotary speed (r/min), D = Drilling diameter (inches), E = Pulldown (feed) force (thousands of pounds per inch of diameter), K = Constant of the rock formation (Table 4.4).

In Fig. 4.21, the drilling energy per unit of volume is represented as function of the compressive strength.

Necessary pulldown force

The power necessary for the pulldown is low when compared to that of rotation, Fig. 4.22.

Besides, the pulldown on the bit, as seen before, depends upon the diameter and the compressive strength of the rock, Fig. 4.23.

The feed capacity of the machine should be 30% more than the maximum working feed.

Once this design parameter is known, the weight of the machine will be defined, as the feed is usually 50% of the working weight, with a 10 to 15% in reserve to assure rig stability during operation and movement.

Compressor

There are two most commonly used types of compressors nowadays: sliding-vane or rotary-screw type. The range of capacities goes from 7 to 70 m^3/min approximately. Both the size and type of this equipment is optional in most cases.

Rotary-screw type compressors work at higher pressure than those of sliding-vane type, have a more simple design and are more compact and give higher mechanical availability.

Type of rolling cone or tricone bit

One of the most important aspects of rotary drilling is the choice of the proper bit. If not, the following will occur:

- Penetration rate lower than optimum.
- Reduced bit life, giving a higher cost per meter drilled.

Table 4.4.

Rock	Compressive strength (MPa)	Constant (K)
Very soft	–	14.10^{-5}
Soft	–	12.10^{-5}
Medium soft	17.5	10.10^{-5}
Medium	56.0	8.10^{-5}
Hard	210.0	6.10^{-5}
Very hard	476.0	4.10^{-5}

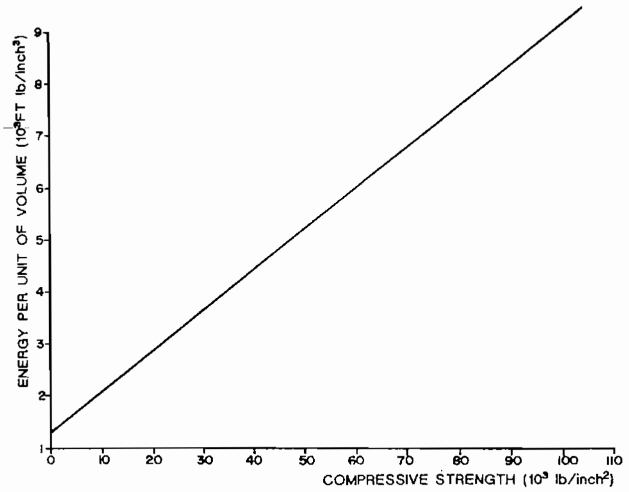


Fig. 4.21. Drilling energy as function of the compressive strength.

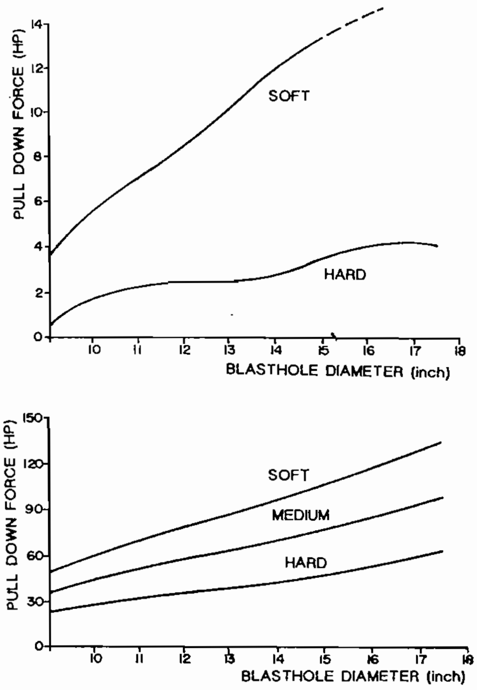


Fig. 4.22. Comparison of rotary power and pulldown force for different types of Rock.

4.12 PENETRATION RATE

Penetration rate depends upon many external factors: geologies, physical properties of the rocks, distribution of stresses and internal structure. This makes it difficult for the project engineer to determine the drilling rate during the actual planning of the project, but highly necessary as the decision taken will effect the rest of the operations.

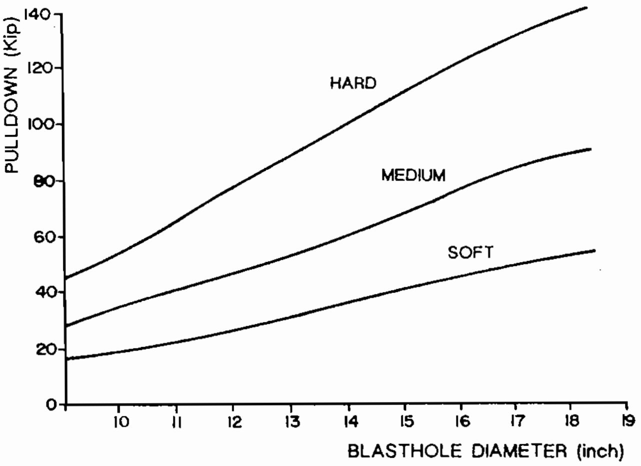


Fig. 4.23. Pulldown force on the bit as function of the diameter and type of rock (1 Kip = 4.448 kN).

There are two procedures that can be used for determining the penetration rate:

1. The taking of representative samples and carrying out scaled down tests by the bit manufacturers. They will then send a report in which the following will be indicated:

- The type of roller tricone recommended.
- Suggested thrust load and rotary speed.
- Estimated penetration rate, and
- Predicted service life of the bit.

The reliability of the results depends on the representivity of the samples sent to the manufacturer and, in general, the reports are usually conservative in their production and cost calculations because the effects of discontinuities and their infilling are not taken into account.

2. Calculation of the penetration rate from the compressive strength of the rock.

This procedure is based upon use of empirical formulas suggested by several investigators.

4.12.1 *Tests on samples*

There are two procedures known as Microbit and Indenter Test.

The first, and oldest, has been used since the decade of the fifties. In consists of drilling small formation samples with a 1¼″ (32 mm) microbit with a 200 pounds load on the bit and at 60 r/min.

Each test is carried out by drilling a hole 3⁄32″ (2.4 mm) deep and recording the drilling time each 1⁄32″ (0.8 mm). After each test the bit wear is measured and registered.

The results obtained are correlated with real data, Fig. 4.24, and the life of the tricone bit is estimated.

The system gives good results in soft to medium formations, that can be drilled with steel tooth roller bits, but in hard formations where tungsten carbide insert bits have to be used, very low penetration rates are obtained

Photo 4.2. Microbit test.

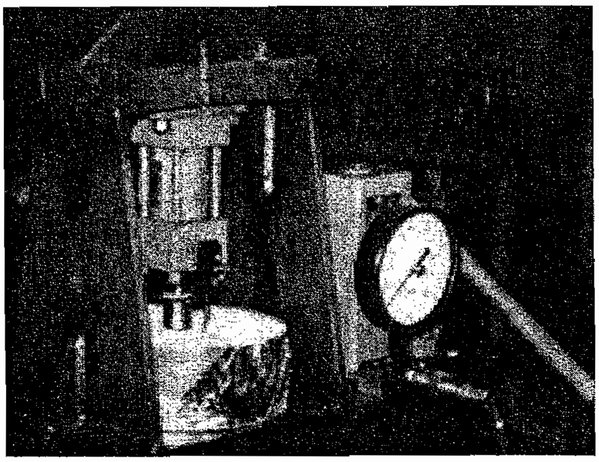


Photo 4.3. Indentor test.

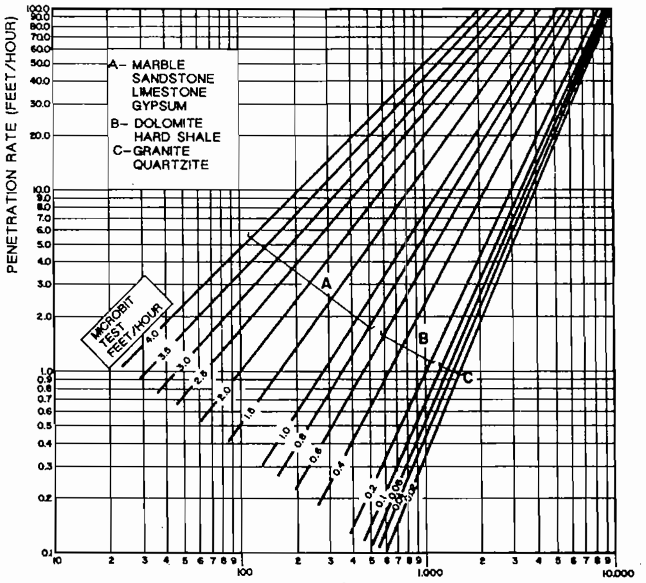


Fig. 4.24. Calculation of the penetration rate from the Microbit tests.

with the microbit and the cutter wear is so high that the results do not give an accurate estimate.

In the second test a hemispherical shaped tungsten carbide tooth or indenter is applied to the sample and the load increased in 500 pounds intervals up to a maximum of 5000 to 6000 pounds, until rock failure occurs, Photo 4.3, establishing the indenter or E_u value which is considered to be the threshold force at time of fracture.

The feed force that must be applied to the bit is:

$$E = E_u \times I_c$$

where: E = Feed force, E_u = Threshold force, I_c = Number of inserts of the tricone that are in contact with the rock at a given instant.

However, I_c usually represents an 8% of the total number C_i of tricone inserts, so the previous equation changes to:

$$E = 0.08 \times E_u \times C_i$$

The penetration rate will be given by:

$$V = N_r \times p$$

where: VP = Penetration rate, N_r = Revolutions per minute, p = Tricone penetration per revolution.

Also, the advance of the tricone bit in a revolution p should be proportional to the penetration p obtained in the test:

$$p = K \times p''$$

where K is a constant that covers the actual field conditions that the test cannot reproduce.

The final equation for penetration rate is as follows:

$$VP = K \times N_r p' = K \times N_r \left(\frac{p'}{E_u}\right) \times E_u = K \times N_r \left(\frac{p'}{E_u}\right) \times \left(\frac{E}{0.08 \times C_i}\right) = K' \times N_r \times \left(\frac{p'}{E_u}\right) \times \left(\frac{E}{C_i}\right)$$

This equation gives the penetration rate from a group of known parameters.

As in most cases the roller tricone bits are thrown away when there is bearing failure, their useful lives can be estimated in meters by multiplying the penetration rate by the number of hours of bearing service life.

4.12.2 *Empirical formulas for estimating the penetration rate*

This procedure is very simple and is based upon empirical formulas that were developed from field tests. They usually take into account the following parameters:

- Drilling diameter.
- Pulldown force on the bit.
- Rotary speed, and
- Uniaxial compressive strength.

The unknown parameter is the Compressive Strength, which can easily be determined by a field or laboratory test, from the Point Load Strength Index.

In 1967, after investigation work was carried out in iron mining exploitations in Canada, A. Bauer and P. Calder suggested the following equation:

$$\log\frac{VP}{6} = K \times \log\frac{E}{\dfrac{RC}{12}}$$

where: VP = Penetration rate (feet/hour), K = Factor that depends on the rock and varies between 1.4 and 1.75 for rocks with compressive strength in the range of 15.000 to 50.000 pounds per square inch, E = Pulldown force (pounds per inch of diameter), RC = Compressive rock strength (pounds per square inch).

In 1971, Bauer modified the equation by introducing another parameter – rotary speed:

$$VP = [61\text{-}28 \log_{10} RC] \times \frac{E}{D} \times \frac{N_r}{300}$$

where: VP = Penetration rate (feet/hour), RC = Compressive strength (thousands of pounds per square inch), E/D = Unitary feed force (thousands of pounds per inch of diameter), N_r = Rotary speed (r/min).

This equation gives good results in the range of compressive strengths mentioned.

In Fig. 4.25, there is a nomograph to calculate the penetration rate as function of the compressive strength.

R. Praillet, in 1978, deducted the following empirical equation:

$$VP = \frac{2.18 \times E \times N_r}{0.2 \times RC \times D^{0.9} \times \dfrac{RC}{10.000}}$$

where: VP = Penetration rate (m/h), E = Feed force (kg), N_r = Rotary speed (r/min), RC = Compressive rock strength (MPa), D = Diameter of the roller tricone bit (mm).

This formula is more reliable in the whole range of compressive rock strengths and permits calculation of the RC value in a working operation.

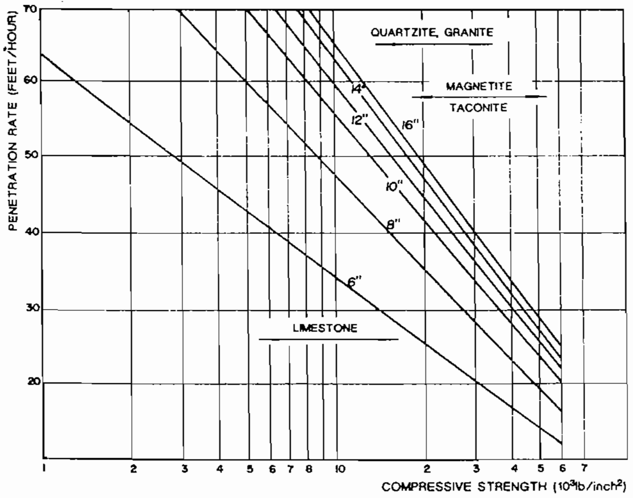


Fig. 4.25. Estimation of the Penetration Rate from the Compressive Strength (Bauer and Calder).

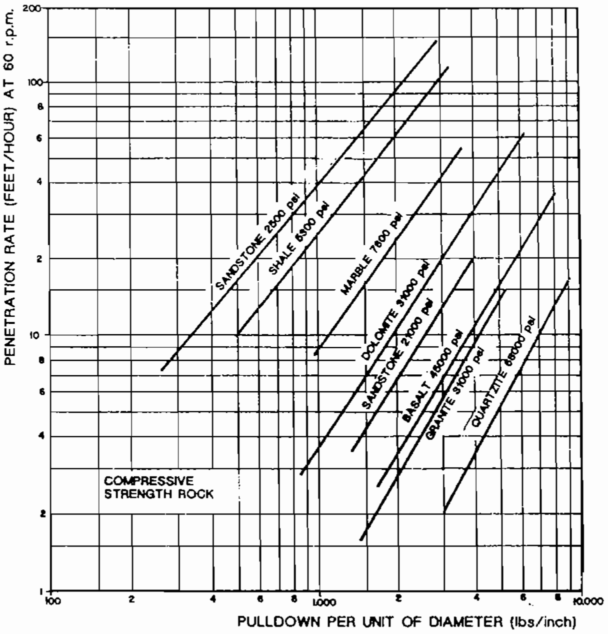


Fig. 4.26. Nomograph of penetration rates.

And lastly, the manufacturers of roller tricone bits have drawn up simple nomographs where, in function with the compressive rock strength, the penetration rate can be calculated for a constant rotary speed of 60 r/min Fig. 4.26.

4.12.3 *Average drilling rate*

Once the penetration rate has been calculated, the average rate should be estimated after including the non-productive times and the mechanical availability of the machine which is usually 80%. It is calculated by the equation:

$$VM = 2 \times VP^{0.65}$$

where: VM = Average drilling rate (m/h), VP = Penetration rate (m/h).

Another more exact way of calculating VM is taking into account the individual non-productive times, discussed before in the chapter on rotary percussive drilling.

4.13 CALCULATION OF DRILLING COSTS

The drilling costs per meter drilled is calculated with the following equation:

$$C_T = \frac{C_A + C_I + C_M + C_O + C_E + C_L}{VM} + C_B$$

where: *Indirect costs*; C_A = Depreciation (\$/h), C_I = Interest rates, taxes and insurance (\$/h). *Direct costs*; C_M = Maintenance (\$/h), C_O = Labor costs (\$/h), C_E = Power costs (\$/h), C_L = Grease and lubrication (\$/h), C_B = Bit, stabilizer and drill pipe (\$/h), VM = Average drilling rate (m/h).

4.13.1 *Depreciation*

The estimated operative life of these machines can be from 50.000 to 100.000 hours for electric rock drills and from 16.000 to 30.000 hours for the diesel-hydraulic units mounted on trucks. In order to calculate the depreciation cost, the purchase price less the residual value is divided by the number of estimated hours.

$$C_A = \frac{\text{Purchase price} - \text{Residual value}}{\text{Hours of operative life}}$$

4.13.2 *Interest rates, insurance and taxes*

Most of the machinery is purchased with borrowed money, thus the interest rates must be taken into account, plus the insurance costs and taxes. In order to calculate the total, the following equation is used:

$$C_I = \frac{\dfrac{N+1}{2N} \times \text{Purchase price} \times \%\ (\text{interest} + \text{insurance} + \text{taxes})}{\text{Work hours per year}}$$

where: N = Number of years of service life.

4.13.3 *Maintenance costs*

This includes repairs due to breakdown and those for preventative maintenance. It can be estimated by multiplying the cost of the machine by 5×10^{-5} for electric rigs, and by 6×10^{-5} for diesel units.

4.13.4 *Labor costs*

This corresponds to the cost/hour of the driller, including social security, vacations, etc., and also that of the helper, when necessary.

4.13.5 *Power costs*

This cost can be electric or diesel energy, and it is calculated from the engine specifications.

4.13.6 *Oil and grease (lubrication)*

This cost is determined from the data supplied by the manufacturer, with reference to the oil changes, hydraulic systems and carter or deposit capacity. The estimate is usually between 15 and 20% of the power costs.

4.13.7 *Average rate*

This is determined from the what was exposed in Section 12 of this chapter.

4.13.8 *Bit, stabilizer and drill pipe*

This is one of the critical areas, as there is a lack of previous information from the technicians, and also for its importance as its influence on drilling costs can run up from 15 to 40% of the total, depending upon the hardness of the rock.

The service life of a tricone bit can be estimated from the following equation:

$$\text{LIFE (m)} = \frac{28.140 \times D^{1.55} \times E^{-1.67}}{N_r} \times 3 \times VP$$

where: D = Diameter (inches), E = Pulldown force on the bit (thousands of pounds), N_r = Rotary speed (r/min), VP = Penetration rate (m/h)

The drill pipes and stabilizers usually have an average life of 30.000 and 11.000 m, respectively.

4.13.9 *Example*

In a mining operation, an electric drill is used with a total pulldown force of 70.000 pounds that drills rock with a compressive strength of 75 MPa, with a 9″ (229 mm) diameter. The actual data of the operation are:

- Feed force on the bit: 39.000 pounds.
- Penetration rate: 34 m/h.
- Rotary speed: 60 r/min

The purchase price of the equipment is 1.85 million dollars, and the administration wants to calculate the costs per lineal meter drilled.

The areas that make up the total cost are:

$$C_A = \frac{1{,}850{,}000\ \$}{80{,}000\ \text{h}} = 23.12\ \$/\text{h (without residual value)}$$

$$C_I = \frac{\frac{13}{24} \times 1.85 \times 10^6\ \$ \times 0.2}{4.300\text{h}} = 46.60\ \$/\text{h}$$

where:

$C_M = 1.85 \times 10^6 \times 5 \times 10^{-5} = 92.50\ \$/\text{h}$,
$C_O = 26.00\ \$/\text{h}$,
$C_E = 185\ \text{kW/h} \times 0.13\ \$/\text{kW} = 24.05\ \$/\text{h}$,
$C_L = 0.2 \times C_E = 4.81\ \$/\text{h}$,
Indirect Costs $= C_A + C_I = 69.72\ \$/\text{h}$,
Direct Costs $= C_M + C_O + C_E + C_L = 147.36\ \$/\text{h}$,
$VM = 2 \times 34^{0.65} = 19.8\ \text{m/h}$.
Life of the roller tricone bit

$$\frac{28{,}140 \times 9^{1.55} \times 39^{-1.67}}{60} \times 3 \times 34 = 3.174\ \text{m}$$

$$C_{B1} = \frac{4{,}750}{3{,}174} = 1.496\ \$/\text{m (Tricone bit)}$$

$$C_{B2} = \frac{2{,}820}{11{,}000} = 0.256\ \$/\text{m (Stabilizer)}$$

$$C_{B3} = \frac{3{,}230}{30{,}000} = 0.108\ \$/\text{m (Drill pipe)}$$

$$C_B = C_{B1} + C_{B2} + C_{B3} = 1.86\ \$/\mathrm{m}$$

$$C_T = \frac{69.72 + 147.36}{19.8} + 1.86 = 12.82\ \$/\mathrm{m}$$

REFERENCES

Bauer, A.: *Drilling and Blasting*. Bucyrus Erie. 1979.

Bernaola, J.: *Predicción de la Velocidad de Perforación a Percusión mediante Ensayo de Laboratorio sobre Muestras de Roca*. Tesis Doctoral E.T.S. Ingenieros de Minas de Madrid. 1984.

Chitwood, B. & Norman, N.E.: *Blasthole Drilling Economics: a Look at the Cost Behind the Cost*. E/M.J. June, 1979.

Chugh, C.P.: *Manual of Drilling Technology*. 1985.

Herranz, F.: *Perforación Rotativa en Minería a Cielo Abierto*. N.P. 1973.

Lauwell, G.W.: *The Pro's and Con's of Rotary Blasthole Design*. Mining Engineering. June-July, 1978.

Lopez Jimeno, C. y E.: *La Perforación Rotativa con Triconos*. Canteras y Explotaciones. Enero, 1990.

Lopez Jimeno, E.: *Implantación de un Método de Cálculo de Voladuras en Banco*. Tesis Doctoral. E.T.S. Ingenieros de Minas de Madrid. 1986.

Manrubia, F. et al.: *Criterios para la Selección de una Perforadora de Rocas*. Industria Minera. 1986.

Martin, J.W. et al.: *Surface Mining Equipment*. 1982.

Morales, V.: *Mining Engineering Handbook – Drilling Machines*. AIME, 1973.

Nelmark, J.D.: *Large Diameter Blasthole Drills*. Mining congress Journal, August, 1970.

Nelmark, J.D.: *Blasthole Design & Applications*. AMF, February, 1983.

Nelmark, J.D.: *Inclined Hole Drilling with Large Blasthole Drills Can Sometimes be Used*. Mining Engineering. September, 1984.

Pla, F. et al.: *Curso de Perforación y Voladuras*. Fundación Gómez-Pardo. 1978.

Praillet, R.: *Consideraciones de un Fabricante de Máquinas de Perforación*. Canteras y Explotaciones. Septiembre, 1984.

Williamson, T.N.: *Surface Mining – Rotary Drilling*. AIME, 1968.

CHAPTER 5

Rolling cone rock bits

5.1 ROLLING CONE ROCK BITS

Although these bits, as drilling tools, made their first appearance in 1910, it can be said that until the development of rotary rigs in the sixties, an improvement in design and manufacturing of this type of bit did not become a reality, bringing about its massive use in mining.

In the beginning the bits were only used in soft formations or those that offered little resistance, but at present, these tools have given rotary drilling a competetive place amongst other hard rock drilling methods.

The work of a rolling cone, or tricone, rock bit is based upon a combination of two effects or actions:

Indentation
The steel teeth or tungsten carbide inserts of the bit penetrate into the rock owing to the pulldown or feed force exerted on it. This mechanism is equal to rock crushing.

Cutting
The fragments of rock are formed from the lateral scraping movement of the cones when they rotate against the blasthole bottom.

The cutting action is only produced as such in soft rocks, as it is actually a complex combination of crushing and shearing owing to the movement of the bit.

5.2 MAJOR COMPONENTS AND DESIGN FEATURES

The main components of a rolling cone rock bit and, in consequence, design features are: the cones, the bearings and the bit body.

5.2.1 *Cones*

The design parameters of the cones are discussed below.

Journal or pin angle of the cone
One of the most important aspects that must be taken into consideration when designing a roller tricone bit is the angle that the center line or axis of the bearing pin and cone make with the horizontal plane. Another explanation would be the angle formed by the axis of the rock bit and a line perpendicular to the axis of the journal. This angle determines the diameter of the Milled tooth cone in accordance with the diameter of the blasthole. If the journal angle is increased, the diameter of the cone should decrease and vice versa, Fig. 5.2.

Fig. 5.3 shows the geometric parameters that characterize the placing of the 'toothed' cones in two types of rock formations.

The advance of the roller tricone bit in the bottom of the blasthole is mostly regulated by the size and shape of the cones, i.e. its profile.

Cone offset
Another design factor to take into account is the offsetting of the cone axes so that they do not meet at the center of bit rotation Fig. 5.4. Cone offset is the parallel or angular horizontal displacement of the center of a cone relative to a vertical plane through the axis of the bit and in the center of the corresponding 120 degree sector. This produces additional scraping action, when tailored to the formation, and in most instances improves penetration rates.

When working in hard rock, the offset is practically null, as the cones have 'true-rolling action' meaning that the cones roll in a true circle with little scraping action. The rock is broken primarily by chipping and crushing. In soft rocks the tendency is towards more offset, which produces breakage through the twisting and tearing action of the teeth due to cone slippage, which scrapes and gauges the bottom of the hole, along with rotation. In medium rocks both effects are combined, rotation and scraping, breaking the rock by crushing and scraping.

Cone angle
The cone angle is inversely proportional to the journal angle, so that when the latter increases, the cone angle should diminish to avoid interference between cones, Fig. 5.5.

Tooth length
In a steel tooth rolling cone bit, the tooth length is determined by the depth of the grooves or milling cut into the cones which form the inner and outer ends of the cone teeth. If the bit has inserts, then the length will be given by the visible part of the hard metal buttons, Fig. 5.5.

Cone shell thickness
There must be a minimum thickness to insure the structural strength of the cone. The thickness is determined by

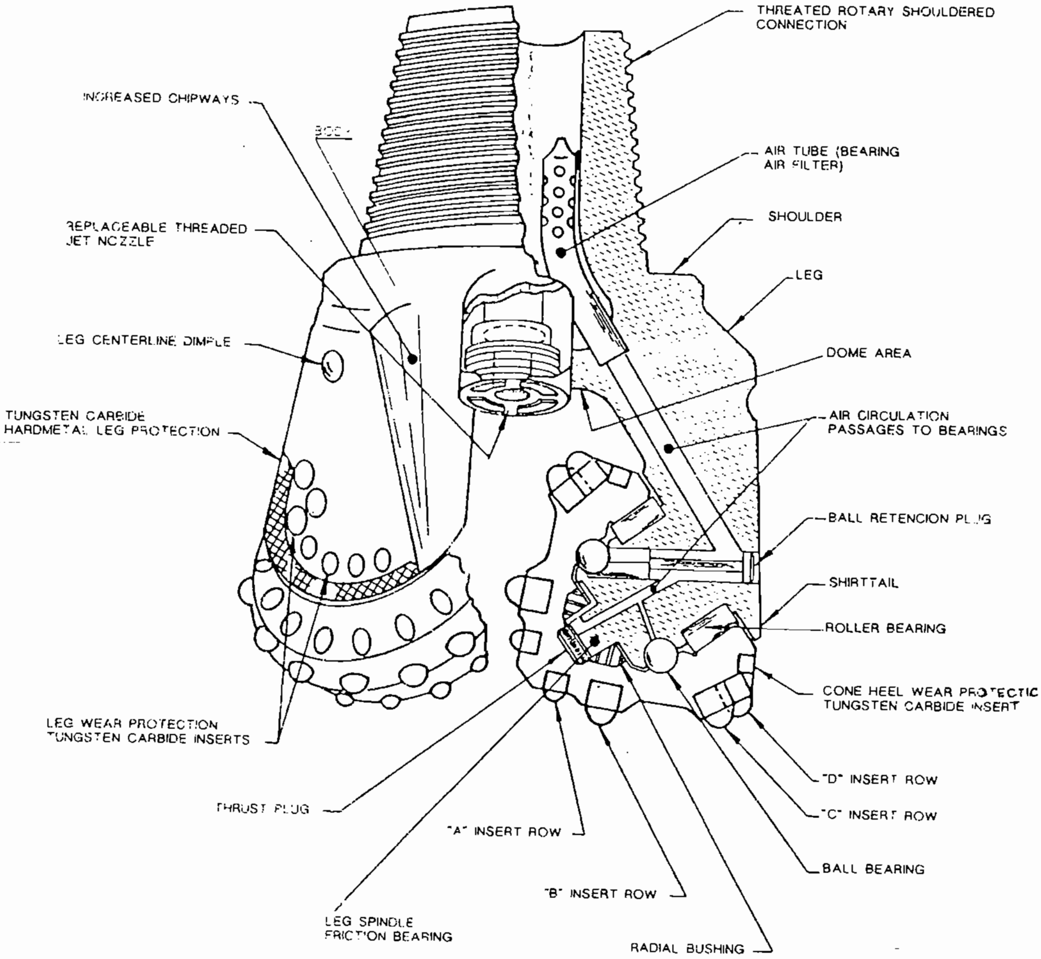

THREATED ROTARY SHOULDERED CONNECTION
INCREASED CHIPWAYS
BODY
AIR TUBE (BEARING AIR FILTER)
REPLACEABLE THREADED JET NOZZLE
SHOULDER
LEG
LEG CENTERLINE DIMPLE
DOME AREA
TUNGSTEN CARBIDE HARDMETAL LEG PROTECTION
AIR CIRCULATION PASSAGES TO BEARINGS
BALL RETENCION PLUG
SHIRTTAIL
ROLLER BEARING
CONE HEEL WEAR PROTECTION TUNGSTEN CARBIDE INSERT
LEG WEAR PROTECTION TUNGSTEN CARBIDE INSERTS
"D" INSERT ROW
THRUST PLUG
"C" INSERT ROW
"A" INSERT ROW
BALL BEARING
"B" INSERT ROW
LEG SPINDLE FRICTION BEARING
RADIAL BUSHING

Fig. 5.1. Elements of a tungsten carbide insert rolling cone rock bit.

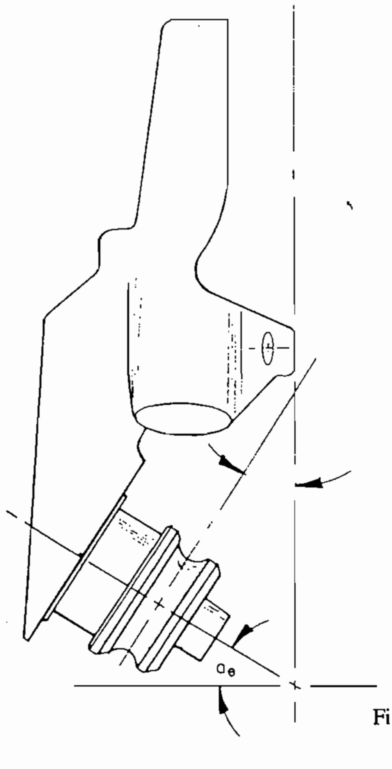


Fig. 5.2. Journal or pin angle.

bearing size, by the depth of the grooves in the steel tooth bits and by the depth of the sockets in the button bits, Fig. 5.5.

5.2.2 *Bearings*

The following are the types of bearings used in tricone bits:

- Roller and ball.
- Flat bearings, also called friction or nose bearings, with lubrication.

The bearing pin roller race is subjected to the major part of the radial load in the cone, while the nose bearing only takes a small part.

The thrust surface that is perpendicular to the guide pin and to the thrust plug is designed to withstand outward loads. The ball race keeps the cone working and is subjected to inward thrust. When other parts of the bearing are worn, the ball race also will withstand some outward radial loads.

In blasthole drilling tricone bits, a high percentage of air is deviated through the bearings in order to cool and clean their elements. By adding oil to the compressed air conduct, bearing life is lengthened and, therefore, the drilling costs lowered.

5.2.3 *Bit body*

The body of the bit is composed of three identical parts that are globally known as head. Each head contains an integral bearing upon which the cone is inserted as well as a deposit and the conduct through which the drilling fluid

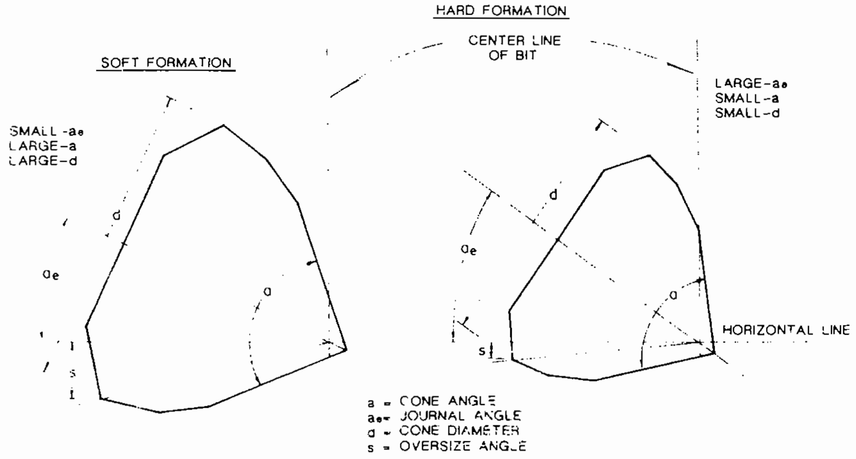


Fig. 5.3. Journal angles in two types of rock.

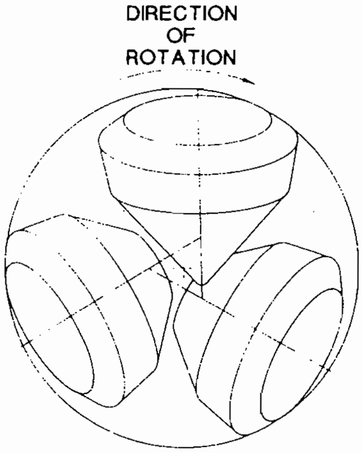


Fig. 5.4. Cone offset (Hughes Tool Co.).

circulates to clean the cuttings from the blasthole bottom.

One of the jobs of the bit body is to direct the flushing fluid towards the area where the cleaning will be most effective.

Modern rolling cone bits have jet nozzles that send the air through the cones straight to the blasthole bottom, thus requiring suitable flow and pressure from the compressors in order to clean the blasthole and cones as well.

The three heads are joined to form a unit by computer controlled welding, later mechanizing the thread where the air tube is inserted.

The thread transmits the torsion and axial forces produced by the rock drill to the bit through the conducts.

5.3 THE METALLURGY OF ROLLING CONE ROCK BITS

One of the most successful achievements in the manufacturing of rock bits has been the use of special alloys that are different for each bit element, Table 5.1.

Table 5.1.

Drill bit elements	Required properties	Type of steel
Cone	Impact and abrasion resistant	Carbon, magnese, nickel and molybdenum
Heads	Fatigue resistant. High impact resistance and weldable	Carbon, manganese, chromium and molybdenum
Roller and ball bearings	High impact resistance	Carbon, manganese, nickel, chromium and molybdenum
Pilot bushing and pilot pins	Wear resistance	Carbon, tungsten, chromium, molybdenum & vanadium
Surface of bearings	Wear resistance	Cobalt, chromium, carbon, tungsten, and nickel
Teeth	High abrasion resistance	Tungsten and carbon
Inserts	High abrasion and impact resistance	Tungsten, carbon and cobalt

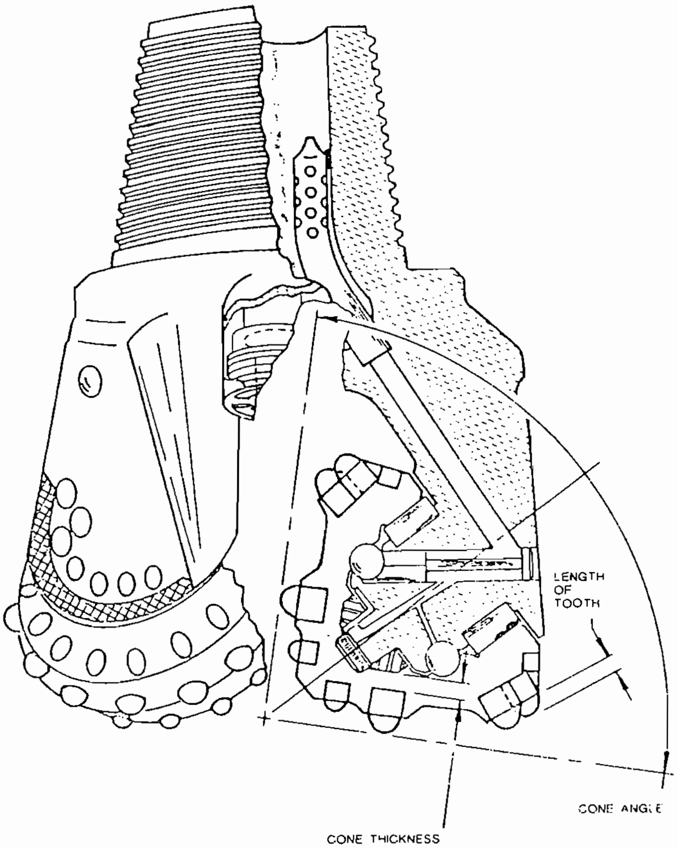


Fig. 5.5. Cone angle, tooth length and cone shell thickness (Smith-Gruner).

5.4 TYPES OF ROLLING CONE BITS

There are two types:

- Steel tooth bits (Milled tooth).
- Tungsten carbide insert bits

The steel tooth bits have the advantage of their low cost, as they are one fifth the price of an insert bit. However, the benefits of the insert bits are:

- They maintain penetration rate during service life.
- Less feed force is required to obtain a penetration rate.
- Lower rotary torque, thus lowering the stresses on the rotary heads.
- Reduced vibrations, producing less fatigue in the rock drill and drill steel.
- Less wear on the stabilizer and the drill pipe because the carbide inserts maintain the diameter of the bit better than the tooth inserts.
- Less non-productive times due to bit changes and less damage to the threads.

5.5 BIT TYPE SELECTION

When choosing a rolling cone rock bit, the compressive strength of the rock and its hardness are the principal factors to take into consideration. Normally, the users send samples to the bit manufacturers for advice on the type of tricone bit necessary, the probable penetration rates and wear resistance in meters.

5.5.1 *Steel tooth rolling cone bits*

The tooth bits are classified in three categories according to the type of rock formation: soft, medium and hard.

Soft formations

The roller cone rock bits for soft formations have small bearings that are compatible with the long teeth and the low feed forces on the bit. The teeth are widely separated and the cones have a large offset to produce a high gouging, scraping effect, Photo 5.1.

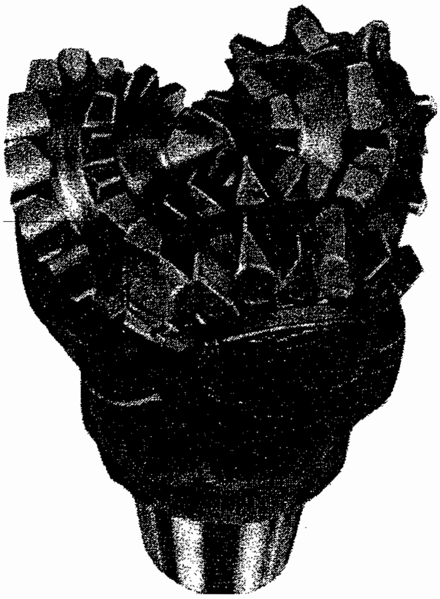


Photo 5.1. Steel tooth roller cone rock bit for soft formations (Hughes Tool Co.).

Medium formations

The bits for medium formations have medium sized bearings in accordance with the necessary pulldown force and tooth size.

The spacing, offset and length of the teeth are smaller than in the bits for soft formations, Photo 5.2.

Hard formations

The roller cones for hard formations have large bearings, and short, resistant teeth that are very close to each other. The cones have very little offset in order to increase penetration by crushing, with very high feed loads, Photo 5.3.

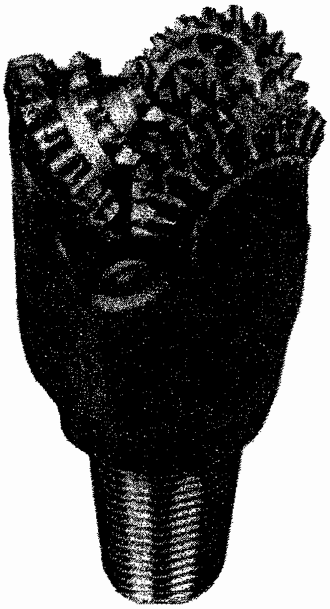


Photo 5.2. Steel tooth Roller tricone for medium formations (Hughes Tool Co.).

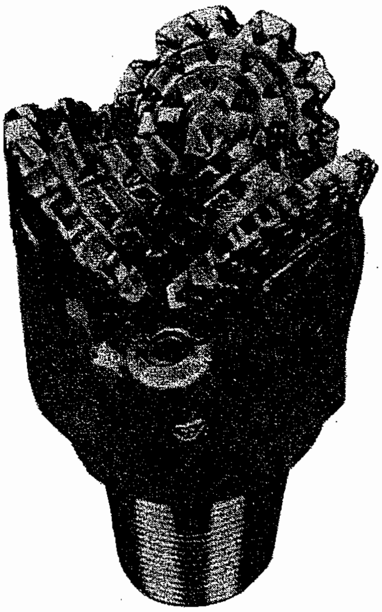


Photo 5.3. Roller tricone for hard formations (Hughes Tool Co.).

Table 5.2. General classification of rolling cone bits.

Rock Formations	Design Features			Cutting Action		Type and Model			
	Tooth spacing	Tooth depth	Tooth hard facing	Chipping crushing	Gouging/ scraping	Hughes	Reed	Smith	Varel
Soft formations Fomations with low compressive strength and high drillability (shale, clay, chalk, marl and sand)	▨	▨	▨	▨	▨	S	TS	GR4	V3 V3M
Medium formations (shale, sandstone, limestone, mudstone)	▨	▨	▨	▨	▨	M	TM	GR1 GR2/2C	V2 VH2
Hard formations (sandstone, silstone, carbonates, shale, limestone, dolomite)	▨	▨	▨	▨	▨	H, HR	TH	GRH GRHC	VH1 VQM

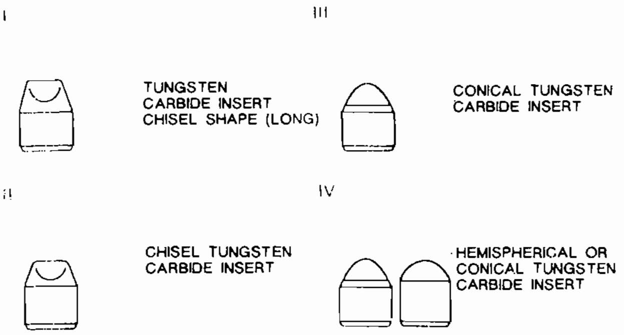

I
III
TUNGSTEN CARBIDE INSERT CHISEL SHAPE (LONG)
CONICAL TUNGSTEN CARBIDE INSERT
II
IV
CHISEL TUNGSTEN CARBIDE INSERT
HEMISPHERICAL OR CONICAL TUNGSTEN CARBIDE INSERT

Fig. 5.6. Tungsten carbide insert shapes for tricone bits.

5.5.2 *Tungsten carbide insert roller tricone bits*

There are four types of tricone bits that are differentiated by their design, size and spacing of the inserts and by the cutting action, Fig. 5.6.

In Table 5.3, the types of rolling cone bits are specified with their cutting action as funtion of the type of rock to be drilled.

5.6 EFFECTS OF THE OPERATING PARAMETERS ON THE ROLLING CONE BITS

The main operating parameters in rotary drilling are:

5.6.1 *Effect of weight on the bearings*

The service life of a bearing is inversely proportional to the cube of the weight placed upon it. However, as in the tricone bits friction elements are used, which suffer fatigue and wear, this relationship is not valid and it is generally accepted that the life of a bearing is inversely proportional to the weight elevated to an exponent that varies between 1.8 and 2.8.

5.6.2 *Effect of weight on cutting elements*

Excessive weight produces breakage of the inserts and wear on the cutting structure in hard rocks.

In soft, non-abrasive formations, the cutting structure rarely limits the life of the bit and high feed load does not produce damage, as long as there is sufficient air to clean the blasthole bottom.

5.6.3 *Effect of rotary speed on bearing life*

Bearing service life is inversely proportional to the rotary speed.

5.6.4 *Effect of rotary speed on the cutting elements*

In abrasive formations, the wear of the inserts increases with rotary speed. In hard formations, a high rotary speed produces breakage on the inserts through impact.

5.7 NOZZLE SELECTION

Rolling cone bits are designed so that a part of the air delivery, approximately 20%, be used for refrigeration and cleaning of the bearings. The rest of the air goes through nozzles to clean the steel teeth of the cones and produce the necessary turbulence to initiate the upward bailing of the cuttings through the annular space. These nozzled have diaphragms which can be placed in different positions to obtain the proper conditions and effectively clean the blasthole bottom. Rechangeable nozzles can also be used for the same objective.

In order to calculate the diameter of the nozzles, for one, or three, the following equations are used:

Table 5.3. General classification of Tungsten carbide insert tricone bits.

Rock Formation	Design Features			Cutting Action		Type of Rolling Cone Bit	Type and Model					
	Insert size	Insert spacing	Insert projection	Chipping crushing	Gouging scraping		Hughes	Reed	Smith Tool	Varel	Security	Atlas Copco
Soft formations Low compressive strength (<40 MPa) and high drillability (talc, shale, clays, gypsum)	▨	▨	▨	▨	▨	I	HH33	M51 M52	Q4JL	QMC9	S8M	CS251 CS311 CS381
Medium to medium hard formations The first with strengths between 40 and 90 MPa (limestone, marbles, schists, fluorites, etc.). The second, with CS between 90 and 170 MPa (dolomites, graywacks, feldspars, granites, gneisses, etc.)	▨	▨	▨	▨	▨	I, III	HH44 HH55	M62 M70	Q5JL	QMC7 QMC77 QMC6	M8M	CM251 CM311 CM381
Hard formations With compressive strengths between 170 and 230 MPa (Quartzites, pyrites, basalts, taconites) more abrasiveness	▨	▨	▨	▨	▨	III	HH77 HH88	M73 M80	Q7JL	QMCS QMCH	H8M H10M	CH251 CH311
Very hard formations Compressive strengths above 320 MPa (lava, topaz, corundum, etc.)	▨	▨	▨	▨	▨	IV	HH99	M83 M84	Q9JL		H10M3 H10M4	CH381

$$d_t = \frac{Q_a}{43.34\,(p_a + 32.4)} \quad \text{for one nozzle}$$

where: d_t = Nozzle diameter (mm), Q_a = Air flow (m^3/min), P_a = Outgoing pressure from the compressor (kPa).

5.8 EVALUATION OF DULL ROLLING CONES

An important job in the effective use of tricone bits is the analysis of the worn out or dull bits, as the identification of possible causes helps to correct errors in operation and improve bit selection. Bit failure is usually due to three causes:

- Bearing failure
- Failure of the cutting structure, and
- Shirttail failure.

a) *Bearing failure*

b) *Cutting structure failure*

5.9 EXAMPLE OF ROLLER TRICONE BIT SELECTION

In an operation, the drilling diameter is to be 9″ (229 mm) with a compressive rock strength of 30.000 pounds/inch^2 (206.8 MPa).

1. The maximum pulldown load on a bit of 9″ is given by the equation:

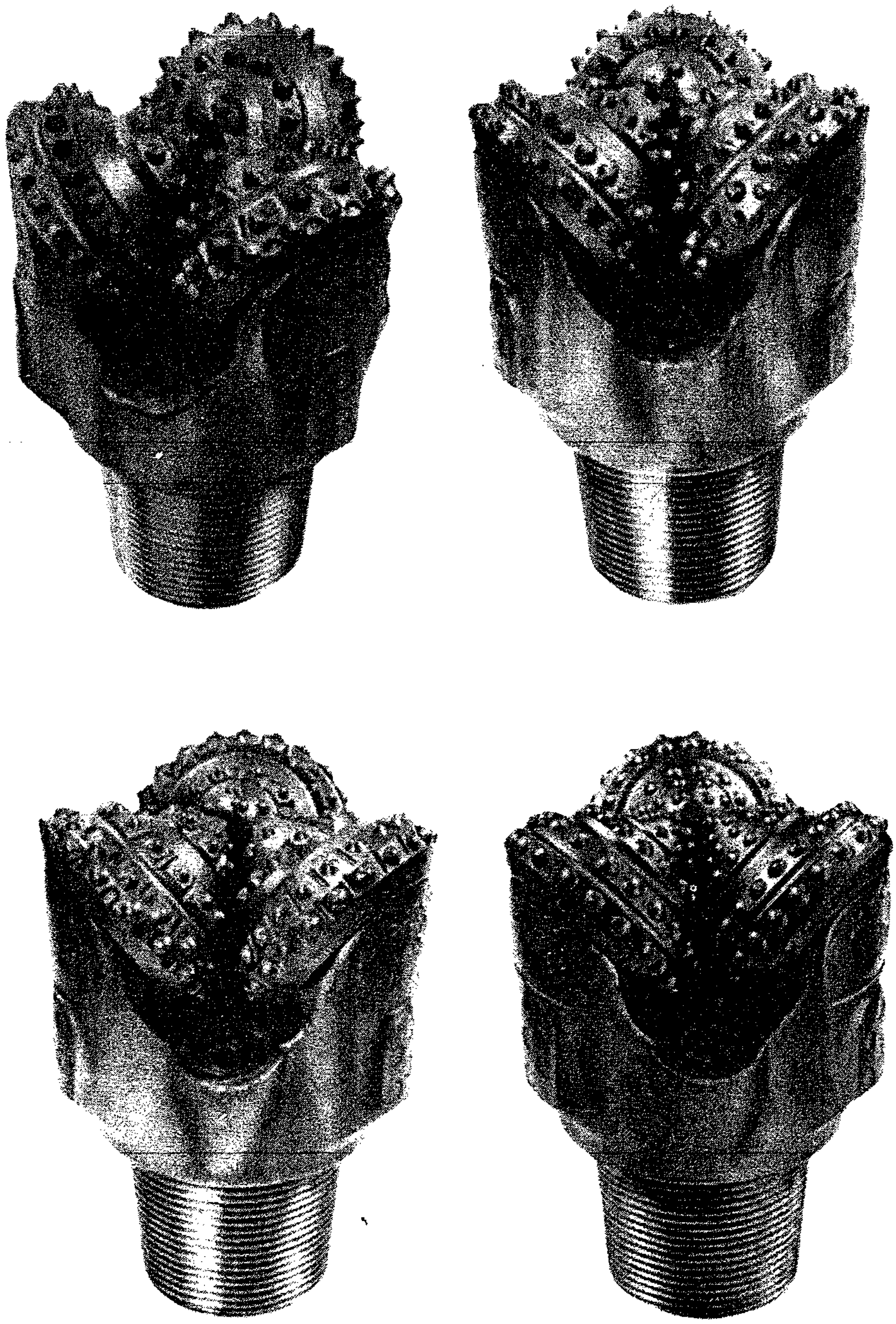
$$E_M = 810 \times D^2 = 810 \times 9^2 = 65.610 \text{ pounds}$$

$$E_i = \frac{65.610}{9} = 7.290\ \frac{\text{pounds}}{\text{inch}}$$

2. The feed load per unit of diameter multiplied by 5 indicates the maximum compressive strength that can be drilled by that bit at maximum feed thrust. In this case we have 7.290 × 5 = 36.450 pounds/inch^2 (251.3 MPa), so the operation can be carried out.

3. The feed load that should be given by the drill is calculated from the compressive strength of the rock and from the diameter:

$$\frac{30.000}{5} \times 54.000 \text{ pounds}$$



Photos 5.4, 5.5, 5.6, and 5.7. Types of inserts for rolling cone bits (Hughes Tool Co.).

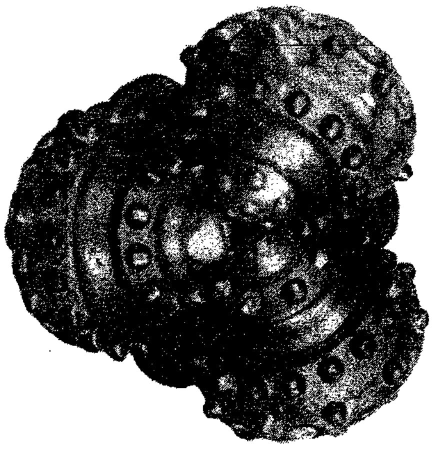


Photo 5.8. Breakage of heel inserts (Hughes Tool Co.).

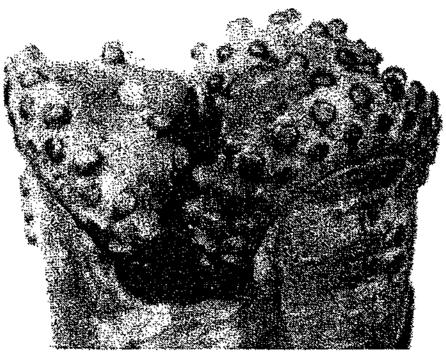


Photo 5.9. Shirttail wear (Hughes Tool Co.).

Table 5.4.

Possible causes	Solutions
Excessive rotary speed	Reduce speed
Wrong type of bit	Change to other type
Insufficient air to cool bearings	Check compressor and drill steel
Blockage in air passage	Check air conduct
Excessive feed load	Reduce load

Table 5.5.

Possible causes	Solutions
Insufficient air to clean center of the blasthole	Increase air volume or reduce advance
Wrong type of tricone	Change to another type
Excessive rotary speed	Reduce speed

Table 5.6.

Possible causes	Solutions
Insufficient air for penetration rate	Increase air volume or reduce advance
Jointed or abrasive formations	Program to thicken or strengthen shirttails
Bending of drill pipe	Change the drill pipe

4. The type of tricone bit is indicated by the whole value that results from dividing the compressive rock strength in pounds/inch^2, by 10.000.

In this example they should by of type III, that is to say, with conic shape tungsten carbide inserts.

5.10 IDAC CODES (International Associationn of Drilling Contractors)

The IDAC Codes is a system of roller tricone rock bit number designation with which the type of bit is specified (steel tooth or tungsten carbide insert), the type of rock formation for which it is intended and some pertaining design features.

First digit (1 to 8)

Steel (Milled) tooth tricone bits:

1-X-X. Soft formations with low compressive strength and high drillability.
2-X-X. Medium to medium hard formations with high compressive strength.
3-X-X. Hard, semi-abrasive formations.
4-X-X. Reserved for future bits.

Insert bits:

5-X-X. Soft to medium formations with low compressive strength.
6-X-X. Semi-hard formations having high compressive strength.
7-X-X. Hard semi-abrasive and abrasive formations.
8-X-X. Extremely hard and abrasive formations.

Second digit (1 to 4)

X-1-X. Numbers 1 to 4 designate formation hardness from softest to hardest within each classification of the series.

Third digit (1 to 7)

Establishes the different features relating to bearings and special design of the gage row (outer row) inserts of the cones. (Common to Milled tooth and insert bits)

X-X-1. Standard tricone with open roller bearings.
X-X-2. Standard tricone with open roller bearings and flushing with air only.
X-X-3. Standard tricone with open roller bearings and special tungsten carbide inserts on the outer heel (gage face) of the cone.
X-X-4. Tricone with sealed roller bearings.
X-X-5. Tricone with sealed roller bearings and special tungsten carbide outer heel (gage face) inserts.
X-X-6. Tricone with sealed friction bearing.
X-X-7. Tricone with sealed friction bearins and special tungsten carbide gage face inserts.

REFERENCES

Anonymous: *Blasthole Technology – Dresser.* Mining Equipment Operation.

Brezovec, D.: *Expensive Bits Are Cheaper to Use.* Coal Age. March, 1983.

Coffman, K.w., And Connors, J.: *Rolling Cutter Bit Development and Application in the Mining Industry. Symposium Materials for the Mining Industry. Colorado, 1974.*

Dresser; Evaluación de las Bocas Gastadas.

Edelberg, V.: *Método de Control Estadístico de los Trépanos Rotary.* Reed International.

Hughes Tool Co.: *Blast hole Bit Handbook. Laboratory Tests Prediction Drillability.* January, 1982.

Maccallum, H. F., and Sanger, J.G.: *Used Rock Bits Tell a Story – Read Them.* Reed Mining Tools.

Reed Tool Co.: *Blast Hole Drilling Technology Handbook. Curso de Formación Técnica sobre Perforación de Barrenos.*

Varel Manufacturing Co.: *Design & Application Data for Varel Drill Bits.*

CHAPTER 6

Rotary drillling with cutting action

6.1 INTRODUCTION

Rotary drilling by cutting action was at its peak in the forties, in American coal mines, for blastholes in overburden and in the ore itself. With growing use in surface operations using rotary rigs with rolling tricone rock bits, this method has been limited to soft rocks, usually with small to medium diameters, clearly competing with direct breakage systems. In underground jobs, rotary percussive drilling has taken over most of the work, leaving only low to medium strength rocks that are non-abrasive (potash, coal, etc.) to the rotary rigs.

Drilling by cutting action in production blastholes is carried out with bits whose structures have elements of tungsten carbide or other materials such as synthetic diamonds or polycrystalines, which vary in shape and angle and can be classified in the following types:

a) Two-wing drag bits, with diameters from 36 to 50 mm.

b) Three and four-wing drag bits with diameters from 50 to 115 mm.

c) Three replaceable blade bit with fluted reamers in diameters that go from 160 to 400 mm.

6.2 FUNDAMENTALS OF DRILLING WITH CUTTING ACTION

The cutting actions of a rotary drag-bit on rock are, according to Fish, the following:

1. Beginning the cycle immediately after the formation of a large fragment, elastic deformations by stresses owing to the angular deflexion of the bit and to torsional strain in the drill rod.
2. Strain energy is released, with consequent impact of the cutting edge against the rock surface, and comminution of rock fragments.
3. Build up of stresses at the bit-rock contact area, with further crushing and displacement of rock debris, until the cutting edge is effectively bearing on a step of unbroken rock which subsequently parts to create a large fragment or chip which, once bailed out, allow a new cycle to start, Fig. 6.2.

The field tests carried out by Fairhurst (1964) show that the pulldown load and the rotary torque upon the bit undergo great variations owing to the discontinuous nature in chip formation, Fig. 6.3.

The cutting force is in function with the geometry of the bit, the compressive strength of the bit and the depth of cut. This force is divided into two, one tangential N_t and another vertical E, Fig. 6.4.

The tangential force is the one that overcomes the compressive rock strength when confronted with the bit. The resisting torque T_r, measured in the axis of the drilling element, is the product of the tangential force multiplied by the radius of the bit. The resisting torque on the total cutting area, supposing that it is a circular crown, is given by:

$$T_r = \frac{2}{3}\mu \times E \frac{r_o^3 - r_I^3}{r_o^2 - r_I^2}$$

where: T_r = Resisting torque, μ = Coefficient of friction, E = Thrust on the bit, r_o = Outside radius of the bit, r_I = Inner radius of the bit.

This resisting torque is determined by the minimum torque of the rock drill that allows the rock to be penetrated. Calling r_e the effective radius of the bit, which is equal to

$$r_e = \frac{2}{3} \times \frac{r_o^3 - r_I^3}{r_o^2 - r_I^2}$$

the previous equation is transformed into

$$T_r = \mu \times E \times r_e.$$

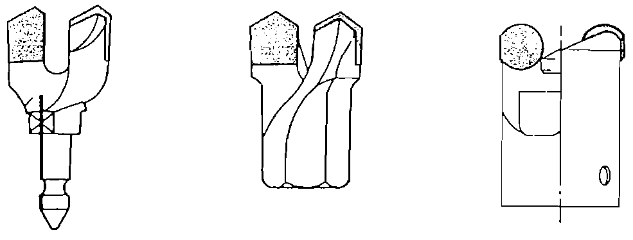
It is deduced that if μ is constant, the torque is proportional to the thrust load on the cutting tool. In reality, the coefficient μ is not constant, as it oscillates with the thickness of the cut and with the feed force itself.

The index that determines the penetration in the rock is obtained by the relationship between the energy consumed by the drill and the specific rock energy. The total energy consumed by the equipment is $2\pi N_r T_r$, where N_r is the rotary speed, which gives the following:

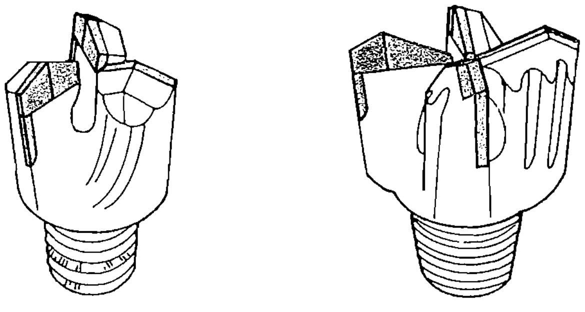
$$VP = \frac{2 \times \pi \times N_r \times T_r}{E_v \times A_r} = \frac{\pi \times \mu \times E \times N_r \times r_e}{E_v \times A_r}$$

where: E_v = Specific rock energy, A_r = Area of the blasthole cross section.

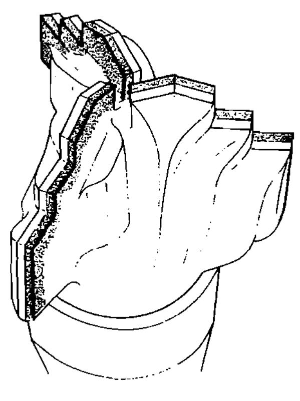
From this relationship it can be deduced that the penetration rate for a given rock and for a determined drilling diameter is linearly proportional to the thrust and rotary speed, although this is not completely true in practice, as it has been indicated that the friction coefficient of the rock varies with the thrust. In Fig. 6.5, it can be observed



a) TWO WING DRAG BIT



b) THREE AND FOUR WING DRAG BIT



c) THREE REPLACEABLE BLADE BIT

Fig. 6.1. Rotary drag bits.

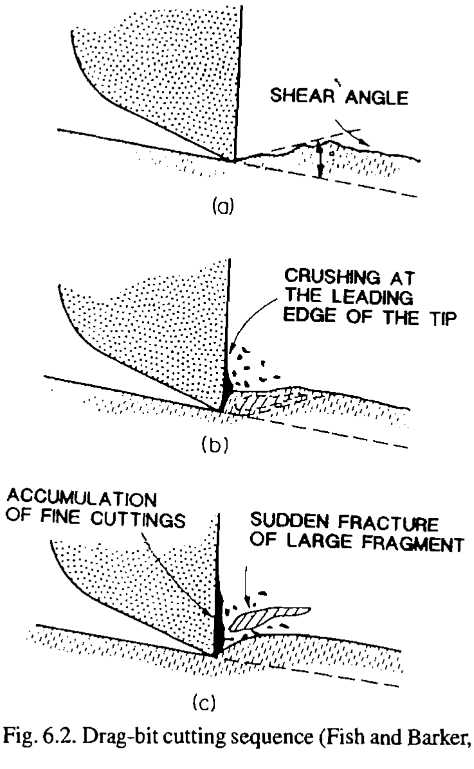

SHEAR ANGLE
(a)
CRUSHING AT THE LEADING EDGE OF THE TIP
(b)
ACCUMULATION OF FINE CUTTINGS
SUDDEN FRACTURE OF LARGE FRAGMENT
(c)

Fig. 6.2. Drag-bit cutting sequence (Fish and Barker, 1956)

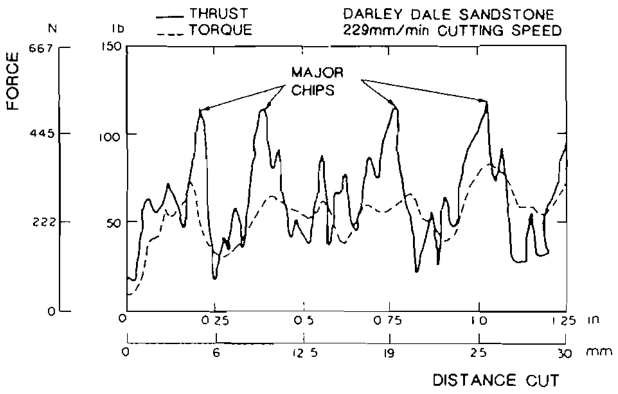


Fig. 6.3. Drag-bit force – displacement curves (Fairhurst, 1964)

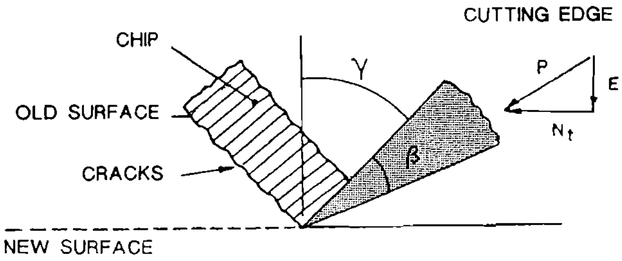


Fig. 6.4. Forces that act upon the cutting tool.

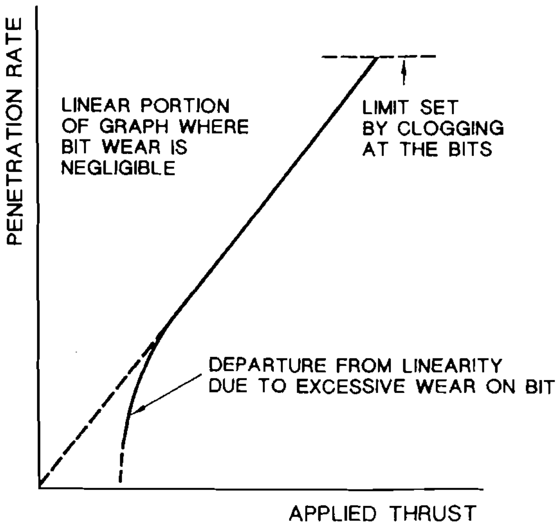


Fig. 6.5. Basic thrust-penetration rate curve for rotary drag-bit drilling (Fish and Barker, 1956).

that there is a thrust value under which a theoretical penetration rate is not achieved, only excessive wear, and a limit value which, if surpassed, will produce clogging of the bit.

The rotary speed is limited by the growing frictional wear on the bits as the number of revolutions increases. Apart from the abrasiveness of the rocks, it must be taken into consideration that the wear increases with higher feed loads and the frictional forces between the rock and bit become higher.

In Table 6.1, the recommended thrusts and rotary speeds are given in function with blasthole diameter and compressive rock strength.

Table 6.1.

Compressive rock strenth (MPa)	Unitary thrust (N/mm)	Blasthole diameter (mm)	Rotary speed (r/min)
< 30	< 140	< 50	< 800
30-50	140-210	> 75	> 100
		< 50	600-800
		> 75	70-100
> 50	> 210	< 50	< 600
> 75	< 70		

Two practical limits of rotary drilling can be given: compressive rock strength, which should be under 80 MPa, and the siliceous content, which should be less than 8%, because, if not, the wear could be uneconomical.

Eimco-Secoma has developed a test for measuring the drillability and abrasiveness of the rocks. It consists of drilling a hole in a rock sample with constant thrust and rotary speed. The bit is of tungsten carbide and the flushing is carried out with water.

A penetration-time curve is obtained and, from this, the drillability index or hardness expressed in 1/10 mm of advance and, by measuring the wear undergone by the calibrated tool during 30 seconds, the abrasiveness is determined in tenths of mm of bit edge wear.

The rocks are clasified in four groups or zones, in function with the two parameters, which define the most adequate drilling methods.

Zone I
Zone with soft formation and low abrasiveness. Dry, low-thrust rotary drilling is suggested with low air pressure.

Zone II
Medium hard formation and low abrasivity. Dry medium-thrust rotary drilling with medium pressure air injection.

Zone III
Fairly hard rock, low abrasiveness. High-thrust rotary drilling and high pressure water flushing. The thrust can reach 20 kN.

Zone IV
Very hard formation and high abrasiveness. Use rotary percussive drilling with air or water flushing.

The drilling parameters for each zone, for drilling diameters between 30 and 51 mm are, according to Secoma, the following:

Zone I
Rotary drilling with little thrust.
- Thrust: From 1 to 8 kN.
- Rotary speed: 800 to 1.100 r/min.
- Dry drilling
- Types of rock: coal, potash, salt, gypsum and soft phosphate.
- Tools: Spiral rods; Two wing drag-bits, $\delta = 110$-$125°$, $\beta = 75°$, $\gamma = 0$-$14°$.
- Drilling rates = 3.5 to 5 m/min.
- With humid air the penetration rates are multiplied by 1.5 and 2.

Zone II
Thrust: 8 to 12 kN.
- Rotary speed: 550 to 800 r/min.
- Drilling with humid air injection.
- Types of rock: Limestone and soft bauxites, soft iron ores.
- Cutting bits: $\delta = 125°$, $\beta = 75$-$80°$, $\gamma = 0$-$2°$.
- Penetration rate: 2 to 3.5 m/min.

Zone III
Thrust: 12 to 18 kN.
- Rotary speed: 300 to 550 r/min.
- Drilling with water injection.
- Types of rock: Bauxites and medium limestones, schists without quartzites, hard gypsums and hard phosphates.
- Cutting bits: $\delta = 125$-$140°$, $\beta = 80°$, $\gamma = -2$-$6°$
- Penetration rate: 1 to 1.8 m/min.

The rotary power, in HP, necessary to make a drag-bit rotate, is calculated with the following equation:

$$HP_r = 8.55 \times 10^{-9} \times D^2 \times N_r \times E^2$$

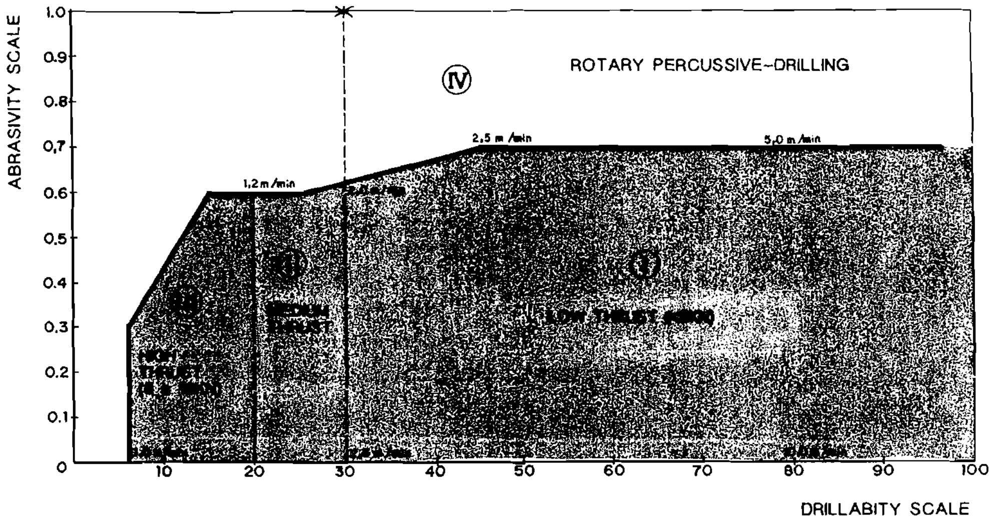


Fig. 6.6. Rock classification according to its drillability and abrasiveness (Eimco-Secoma).

where: D = Diameter (mm), N_r = Rotary speed (r/min), E = Thrust load (kN).

The necessary rotary torque is determined from the equation:

$$T_r = \frac{HP_r \times 7.14}{N_r}$$

where: T_r = Rotary Torque (kN.m).

6.3 FLUSHING OF DRILL CUTTINGS

Drill cuttings are eliminated with a flushing fluid that can be air, in surface operations, or water or humid air in underground jobs.

The advantages that the use of air with water injection brings are the following:

- It facilitates upward bailing, thus increasing the advance rate.
- It cools the drill bit, reducing wear.
- It avoids blasthole filling.
- It eliminates dust which is very important in abrasive formations.

According to Eimco-Secoma, in order to inject humid air around 1.000 to 1.500 l/min of air are necessary and, for each rock drill, about 250 cm^3/min of water.

In very soft rocks, from 30 to 40 MPa, helicoidal drill steel can be used, with larger pitch as the penetration rate increases for efficient removal of the drill cuttings, Fig. 6.7.

In Table 6.2, apart from the typical penetration rates in different types of rocks, the most commonly used flushing systems are indicated.

Table 6.2.

Type of rock	Penetration rate (m/min)	Flushing system
Hard gypsum	1.5-2	Water
Limestone, bauxite	1.5-2.5	Water
Soft iron ore	1.5-3	Water or dry
Soft gypsum	3.8-6	Humid air or dry
Phosphate, coal, salt, potash	3.5-10	Humid air or dry

6.4 CUTTING TOOLS

The cutting efficiency of a tool depends largely upon its design, according to the type of rock that is to be drilled. Fig. 6.8.

The attack angle δ usually varies between 110° and 140°, becoming increasingly obtuse in harder rock: if not, the hard metal would splinter. On occasions bits have been designed with rounded contours.

The angle of the cutting wing β varies between 75 and 80° and that of the cut γ between –6 and 14°, being positive in soft rocks and negative in hard rocks.

Lastly, the backing-off angle or clearance angle is $\delta = 90° - \beta = \gamma$.

During drilling, a point on the cutting bit located at a distance r advances along a helical path. The angle of inclination of this helix is:

$$\omega = \arctan\left(\frac{p}{2\pi r}\right)$$

where p is the advance of the bit per revolution.

Owing to the movement of the bit along the helix, the effective clearance angle is reduced:

$$\varepsilon = \delta - \omega$$

For points near the center of the bit the effective clearance angle is zero, as in these zones the tool compresses the rock. For this reason, drag-bits designed with a central gap usually reach higher drilling speeds.

At the end of the seventies, General Electric manufactured the first Compact Diamond Polycrystalline-PDC,

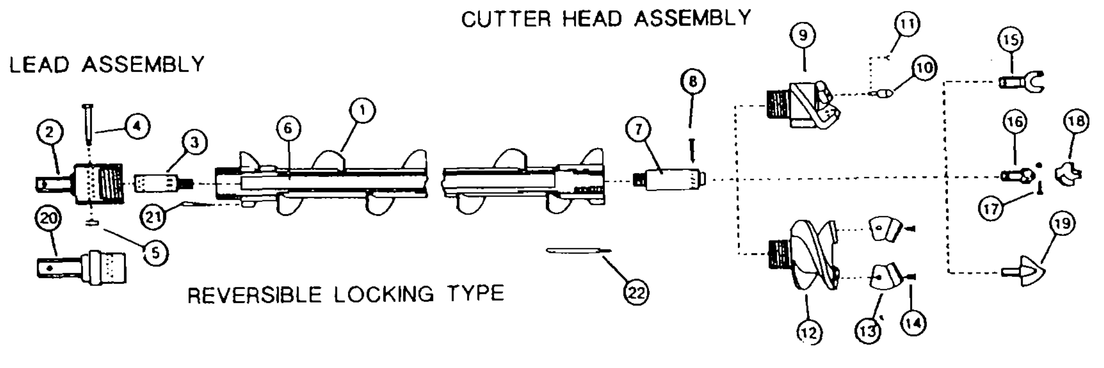


Fig. 6.7. Helicoidal drill rod and bits with different configurations.

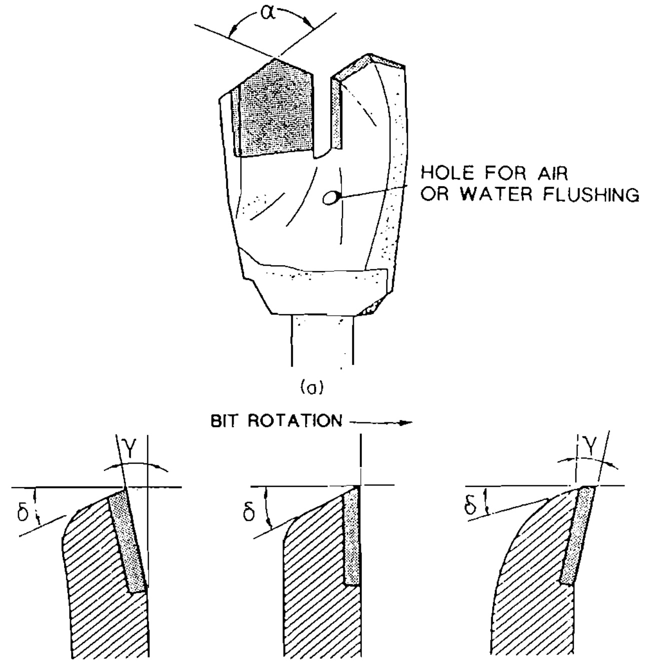


Fig. 6.8. Some characteristics of a cutting tool (Fish and Barker, 1956).

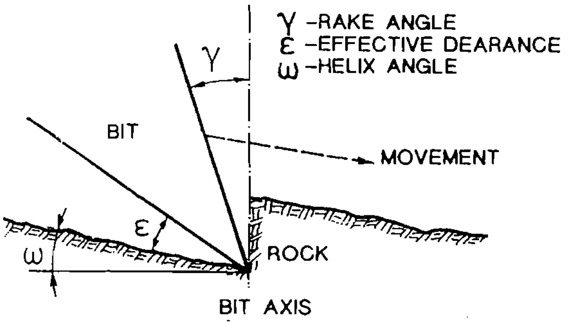


Fig. 6.9. Direction of a point on the the bit (Fairhurst, 1964).

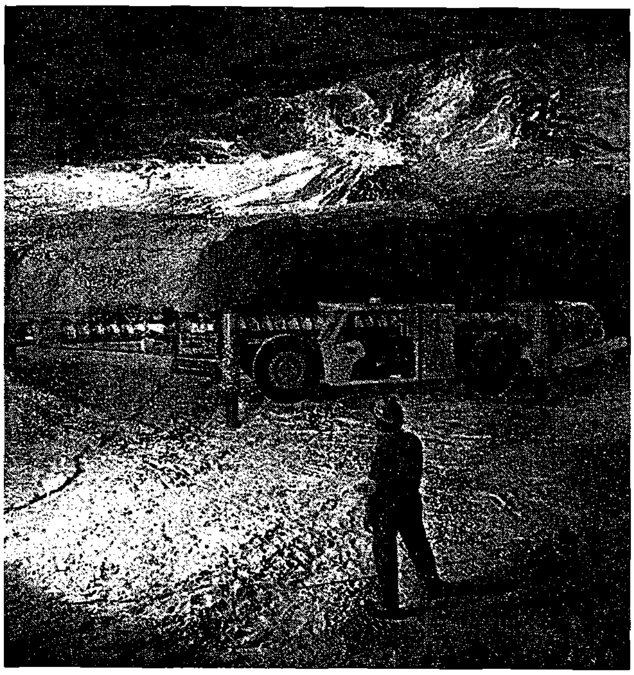


Photo 6.1. Rotary drilling equipment with heicoidal drill steel in a potash mine.

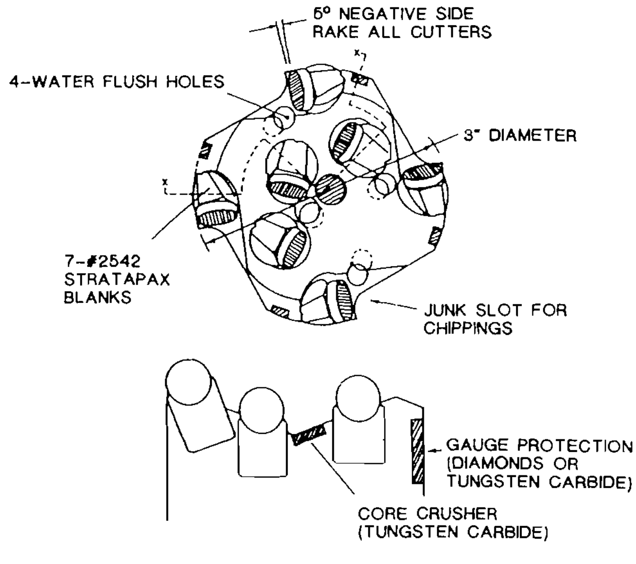


Fig. 6.10. Drill bit with diamond cutting elements.

obtained from a mass of very fine diamond particles that are sinterized under extreme pressure and embedded in tungsten carbide bases that are shaped at high pressures and temperatures. The resulting alloy has exceptional abrasion resistance along with the high resistance of tungsten carbide to impacts.

The present day diamonds are thermically stable up to 1200°C in non oxidizing atmospheres and are available in sizes that range from 0.005 to 0.18 g (0.025 to 0.9 carats) in triangular prism, parallepiped or cylinder shape.

Apart from their use in exploration drilling, diamond bits are used in underground mining for coal, potash, salts and gypsums to drill small diameter blastholes, from 35 to 110 mm.

In many instances, the penetration rates obtained and the service lives of these bits are quite superior to their conventional counterparts.

REFERENCES

Atkins, B.C.: *Drilling Application Successes Using Stratapax Blank Bits in Mining and Construction.* Australian Drilling Association Symposium, 1982.

Bernaola, J.: *Perforación Rotativa.* II Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo. 1987.

Morales, V.: *La Selección y el Funcionamiento de los Triconos.* Canteras y Explotaciones. Septiembre, 1984.

Roberts, A.: *Applied Geotechnology.* Pergamon Press, 1981.

Rodriguez, L.: *Perforación Hidráulica Rotativa en Proyectos Subterráneos.* I Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, 1986.

Tandanand, S.: *Principles of Drilling.* Mining Engineering Handbook. SME. 1973.

CHAPTER 7

Special drilling methods and mounting systems

7.1 INTRODUCTION

Apart from the standard drilling equipment, there are units and mounting systems on the market for special or very specific applications.

Among these jobs, a few can be mentioned such as: drilling rock masses with overburden of a non-consolidated material and/or sheets of water, drilling rigs for shafts and raises, thermal and water jet drilling, etc.

7.2 DRILLING THROUGH OVERBURDEN

These drilling methods were developed to solve problems that appeared when drilling in rocky ground, unconsolidated or alterated masses, overburdens, etc., that require continuous casing tubes to maintain blasthole stability.

Some of the applications for these systems that are in use at present are:

- Drilling for underwater blasting
- Drilling for rock mass blasting with overburden that has not been removed previously.
- Anchoring
- Foundations
- Water wells
- Soil and core sampling, etc.

The overburdens can be beds of natural clay, sand, gravel, etc., as well as of fill with compact or non-compact materials, rock fill, etc.

Drilling can be carried out, as will be noted later on, with top hammer or down-the-hole hammer, and consists of drilling through the overburden at the same time that the casing tube is passed down into the hole, to keep loose material from caving in and blocking the hole, so that drilling can proceed into solid rock.

One important feature of these techniques is that the flushing, or bailing out, of the debris be very effective. It can be carried out centrally through the shank adaptor or through a separate flushing head, in which case the fluid pressure should be higher.

The two methods that have been developed are known as OD and ODEX.

7.2.1 *OD (Overburden Drilling) Method*

In this method, the descent of the casing tube is carried out by percussion and rotation. The equipment consists of an outer casing tube with a tungsten carbide ring bit mounted on the lower end. The casing tube encloses an inner drill string of standard drill steel which is extended by use of coupling sleeves that are independent from those of the casing tube. The casing tubes as well as the drill steel is connected to the hammer by a special shank adaptor, which transmits impact force and rotary force to both, Fig. 7.1.

The basic operations for application of the system are:

- The casing tubes, with or without the inner drill steel, proceed simultaneously through the overbur-den.
- The outer ring bit advances a few centimeters when it reaches the bedrock.
- Drilling is carried out with the inner drill steel unless decomposed or sand beds are encountered, in which case the casing tube would descend at the same time.
- The extension rods are drawn up.
- The plastic casing tubes are allowed to remain in the hole to serve as channels for charging the explosive, or plastic tubes are inserted for this purpose, and
- The casing tubes can be removed.

As between the casing tube and the blasthole walls there is friction which increases with depth, the rock drills should be used with high rotary torque.

Water is usually the flushing fluid in these cases, or compressed air with or without foam. If the upward bailing of the cuttings is insufficient with central flushing, then lateral flushing can be added.

7.2.2 *ODEX Method (Overburden Drilling with Eccentric)*

In this method, based on the principle of underreaming, the casing tube is driven into place by vibrations from the drill and its own weight. Very little rotation is necessary.

The equipment consists of an eccentric reamer bit that drills a hole with a larger diameter than the casing tube which descends as drilling advances. Once the required depth has been reached, the drill string reverses and the reamer bit becomes concentric, loosing diameter, and can then be drawn up through the casing tube. The standard drill steel is then introduced and drilling continues, Fig. 7.3.

The rotary percussive rigs can be top or down-the-hole hammers. If top hammers are used, the percussion impacts are transmitted to the casing tube by means of a driving cap and shank adaptor which make the tube rotate slightly and vibrate. The flushing can be central or lateral, Fig. 7.4.

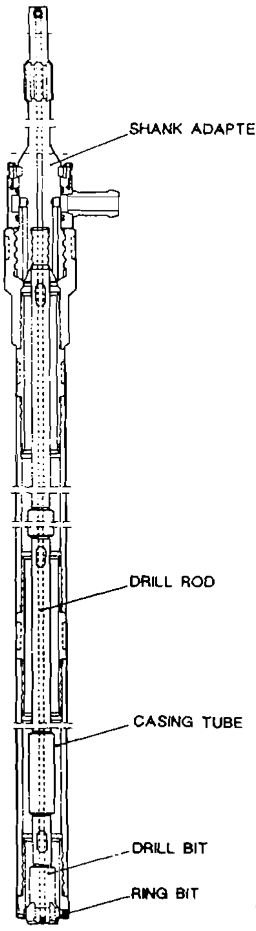


Fig. 7.1. The OD Equipment (Atlas Copco).

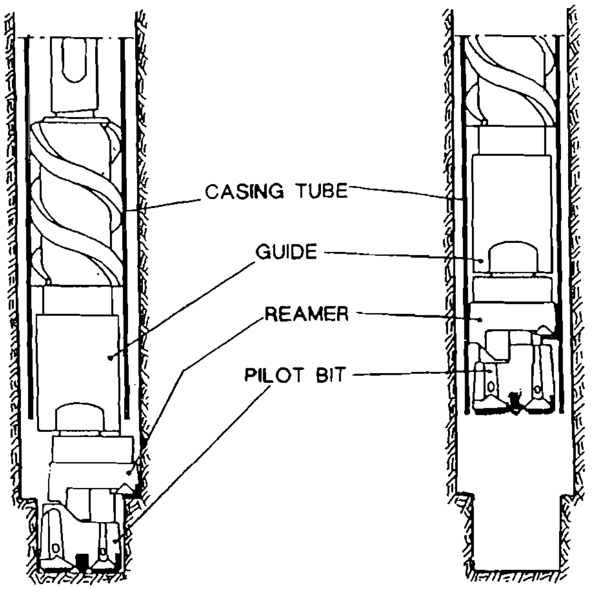


Fig. 7.3. The ODEX method (Atlas Copco).

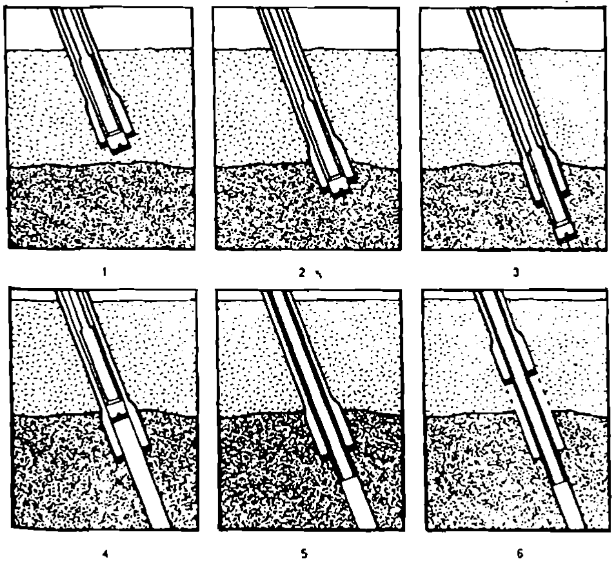


Fig. 7.2. Operations in the OD System.

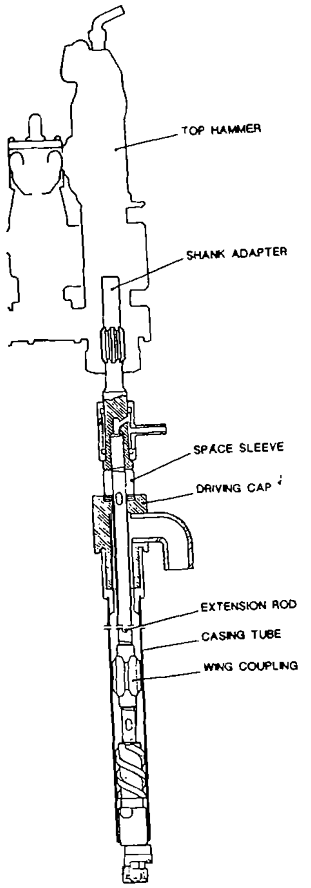


Fig. 7.4. ODEX top hammer equipment (Atlas Copco).

Table 7.1.

Data	For-down-the-hole-drills						For top hammers	
	ODEX 90	ODEX 115	ODEX 140	ODEX 165	ODEX 215	ODEX 76	ODEX 127	OD 72
Min. inside diameter (mm)	90	115	140	165	215	76	127	72
Diameter of reamed hole (mm)	123	152	187	212	278	96	162	108
Normal max hole depth in overburden (m)*	60	100	100	100	100	40	40	40
Inner equipment	3"DTH	4"DTH	5"DTH	6"DTH	7-8"DTH	R38	R38	R38
Casing tube	Weld thread	Weld thread	Weld	Weld	Weld	Weld thread	Weld thread	Thread

*ODEX 90 at 1.2 MPa, ODEX 115-215 at 1.8 MPa. Source: Atlas Copco.

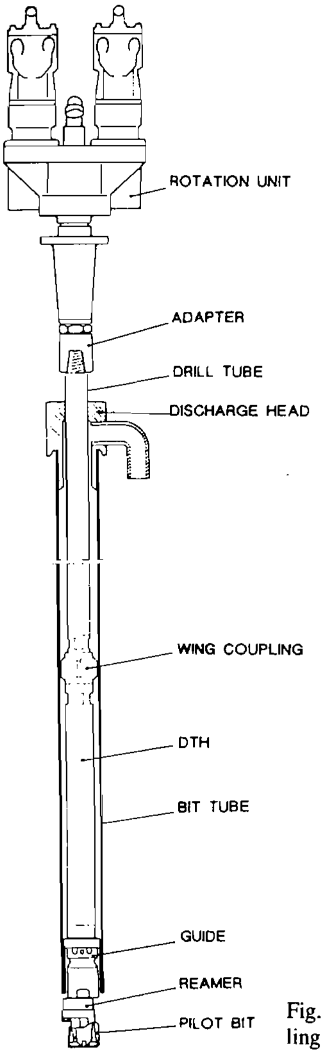


Fig. 7.5. ODEX down-the-hole drilling equipment (Atlas Copco).

When down-the-hole hammer is used, the unit has only one wing coupling, as drill tubes are used instead of extension rods. The string of casing tubes is pulled down by means of a specially designed bit tube, and the flushing is carried out through the rotary head, Fig. 7.5.

In both methods the cuttings are swept upwards through the annulus that remains between the casing and the drill steel, going out through the headstocks.

The flushing fluid can be air up to a depth of 20 m, below which the addition of a foam is recommended to increase the bailing efficiency, wall stability, lower wear and increase penetration rate.

This method has numerous advantages, although some important aspects that should be studied are: the sizes of the casing tubes, the flushing and the drilling system.

The depth of the blastholes must be taken into account when choosing the equipment. In Table 7.1, a selection guide for both drilling methods is given.

On the other hand, as to the applications for these drilling methods, aside from the one described for rock fragmentation blastholes, Table 7.2 indicates other possibilities.

Photo 7.1. ODEX drill bit.

Table 7.2.

	ODEX					OD	
	90	115	140	165	76	127	72
Water well drilling	O	X	X	X			
Road embankment	O	O	O	O	O	O	X
Underwatter drilling					O	O	X
Blasthole drilling	O	O			X		X
Anchoring	X	X			X		X
Injection	X	X			X	O	X
Prospecting	X	X			X	O	X

X = Suitable, O = Can be used. Source: Atlas Copco.

7.3 SHAFT SINKING

When excavating long, large section shafts or metal structures pneumatic or hydraulic jumbos are used with three or four booms with the same number of feeds and rock drills.

When working, these rigs rest on the bottom of the shaft and are anchored to the walls with horizontal hydraulic cylinders. The central supporting column can turn 360°, and the booms, which are similar to the jumbos used for tunnelling, can vary their inclination with respect to the vertical and lengthen themselves if they are telescopic.

Once each round is drilled and charged, the rig is folded and moved to a safe position, later carrying out the mucking operation with twin valve ladles or hydraulic clam shells, as shown in Fig. 7.6.

There are also platforms that have been designed to widen shafts.

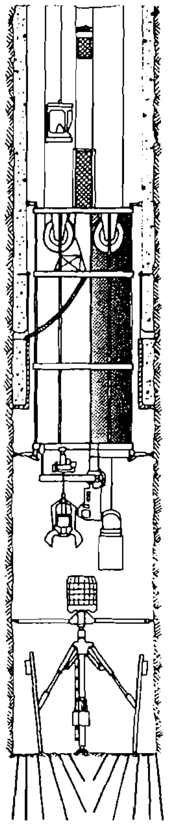


Fig. 7.6. Complete equipment for shaft sinking.

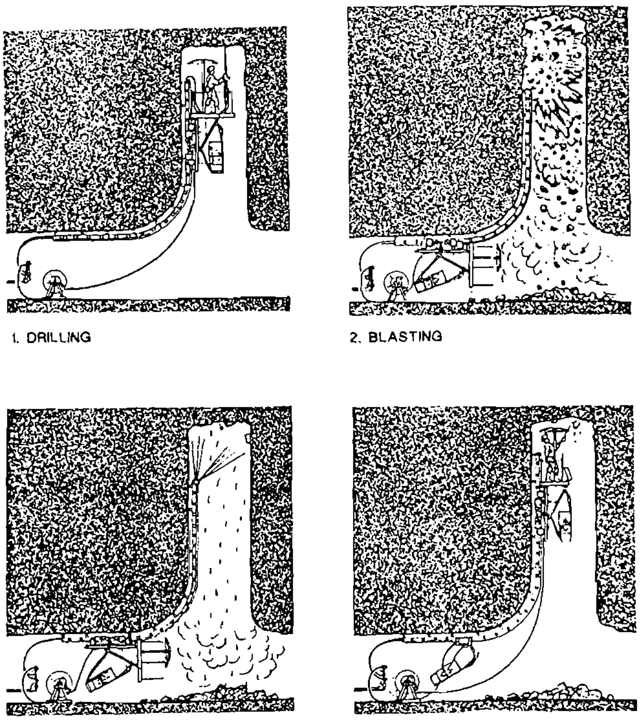


Fig. 7.7. Work cycle with an Alimak platform.

7.4 RAISE DRIVING

7.4.1 *Alimak raise climber*

This excavation method for raise driving was introduced in 1957 and since then, due to its flexibility, economy and speed, it is one of the most widely used in the world, especially in cases when there is no other access to the upper level.

This equipment consists of a cage, the work platform, the driving motors, the guide rail and auxiliary elements. In Fig. 7.7, a complete work cycle is shown.

The platform climbs along a pin rack welded to a guide rail and driven by either compressed air, electric or diesel-hydraulic motors. The guide rail is bolted to the wall with special Alimak design expansion bolts. The air and water pipes, which supply the necessary ventilation and water spray, are placed on the inside of the guide rail for their protection.

During work, the drillers are on a safe platform, as it is covered and has a protective railing. Men ride up to the face safely in the cage, which is under the platform.

In each work shift two drillers can advance from 2.2 to 3 m. Air engines are adequate for lengths under 200 m, the electric for up to 800 m, and from these distances on, diesel-hydraulic engines are recommended.

The main benefits of these rigs are:

– They can be used for raises of any length and inclination.

– Different lengths and geometries of the raises can be achieved by changing the platforms. It is possible to drive cross sections from 3 to 30 m^2, Fig. 7.8.

– In the same operation it is possible to change the direction and inclination of raises by using side-bent (curved) guide rails.

– The length or height of the raises is practically unlimited. Up to the moment, the longest raise driven is 1.040 m long with a 45° inclination.

– It can be used as production equipment in some ore beds by applying the Alimak Raise Mining method, Fig. 7.9.

– The enlarging of pilot raises for excavation of large cross section shafts can be aided by using horizontal drilling units.

– The basic equipment can be used to open various raises simultaneously.

– In poor ground the platforms can be used as supports with bolting, injection, etc.

– The investment is lower than with the Raise Borer System.

– The labor does not have to be highly specialized.

– The initial preparation of the work area is minimum.

On the other hand, there are a few disadvantages:

– Poor quality work environment.

– The walls are very rough which is a problem for ventilation raises and an advantage in ore passage outlets.

– The remaining rock mass is left in poorer condition than with the Raise Boring method.

7.4.2 *The Jora method*

This machine is manufactured by Atlas Copco and can

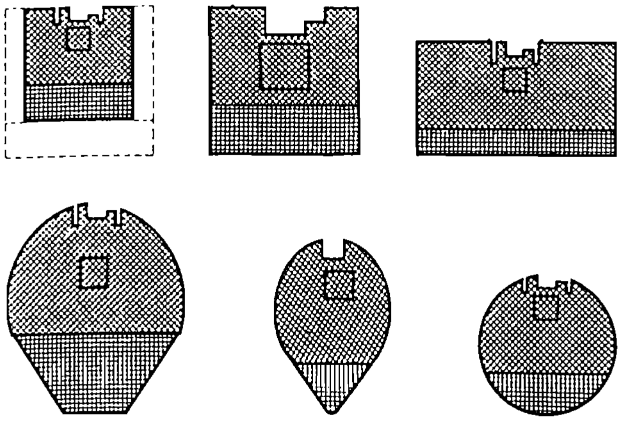


Fig. 7.8. Different platform configurations.

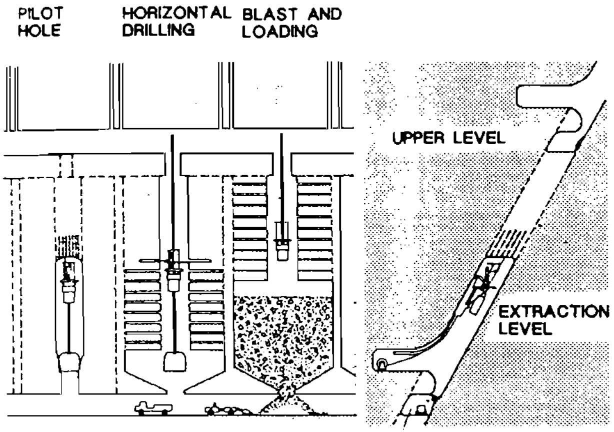


Fig. 7.9. Exploitation method in narrow and inclined beds.

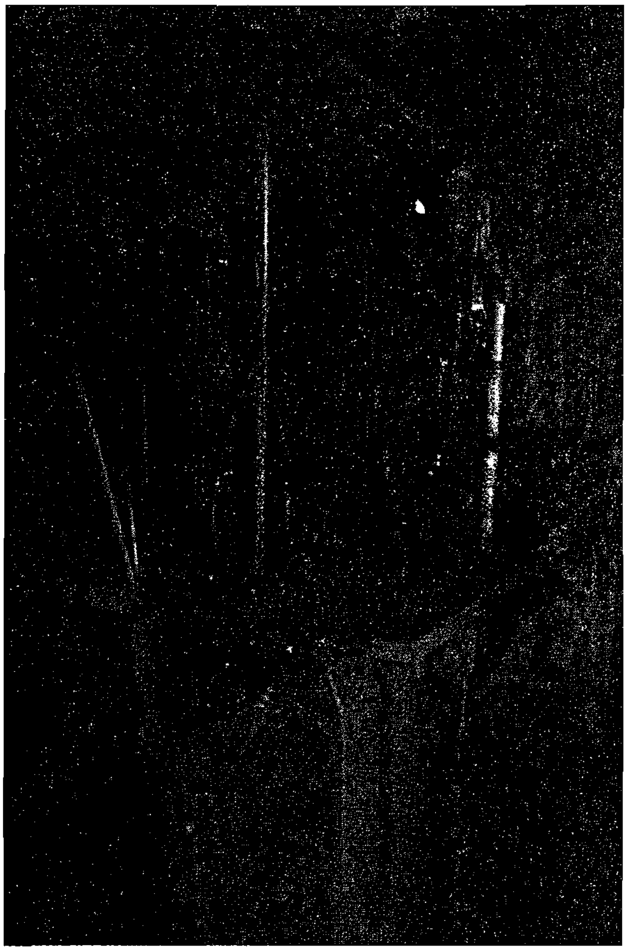


Photo 7.2. Work on Alimak platform.

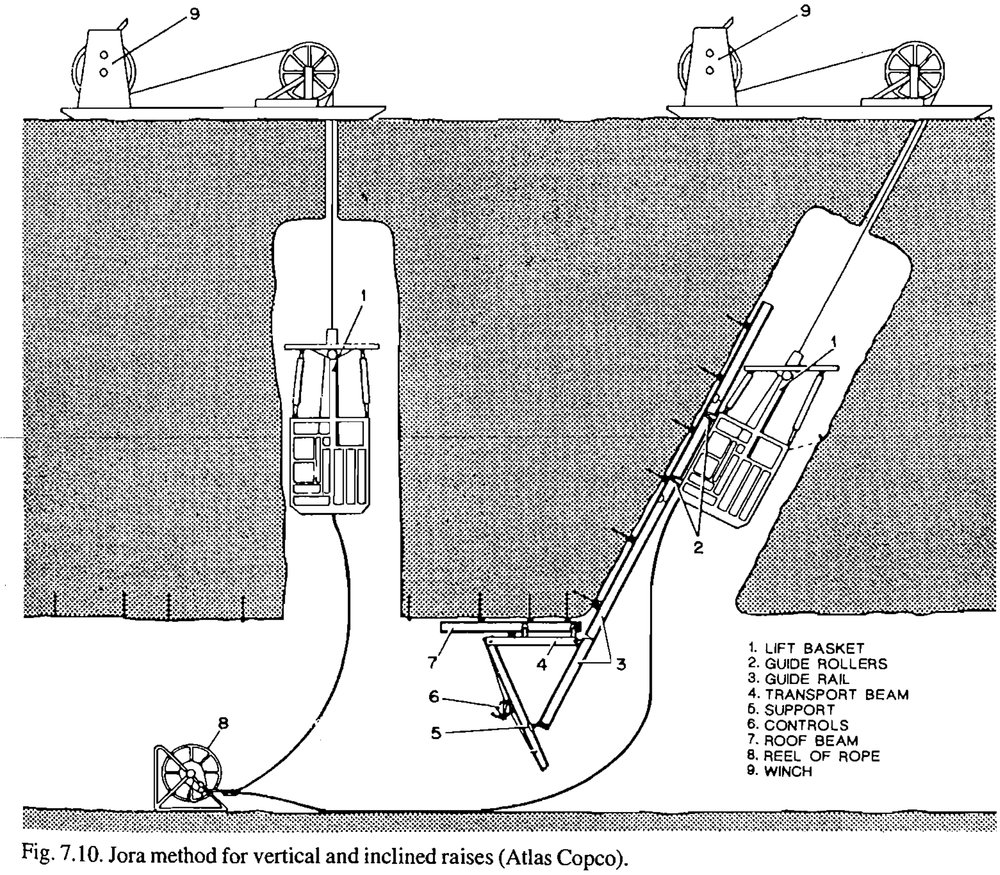


Fig. 7.10. Jora method for vertical and inclined raises (Atlas Copco).

also be used in raising and ore outlets, whether vertical or inclined. The principal difference when compared to the previous equipment is the drilling of a pilot hole with a diameter between 75 and 100 mm through which the cable which holds the lifts is lowered. The main components are the work platform, the lift basket, the hoisting mechanism and, in inclined raises, the guide rail, Fig. 7.10.

During drilling, the platform is anchored to the raise walls by a system of telescopic booms. The main inconvenience of this method, against the former, is the pilot hole drilling, as the maximum raise height will depend upon the accuracy of its alignment. Its practical and economical field of application is between 30 and 100 m.

For each round it is necessary to remove the cage from the hoisting cable, because, if not, the cable would be damaged during blasting. The central blasthole serves as expansion space for parallel cuts, obtaining advances per round of 3 to 4 m, and also as an entrance for fresh air.

7.4.3 *Raise Boring (Full-face) method*

This method, which has become increasingly popular over the past 20 years, consists of the cutting or reaming of the rock with mechanical equipment.

Its main advantages are:

- Excellent personnel safety and good work conditions.
- Higher productivity than in conventional methods of rock breakage with explosives.
- Smooth walls, with minimum losses due to air friction in the ventilation circuits.
- Overbreak does not exist.
- High advance output.
- Possibility of drilling inclined raises although it is better adapted to vertical ones.

The most important disadvantages are:

- Very high investment.
- High excavation cost per lineal meter.
- Lack of flexibility, as the sizes and shapes of the raises cannot be varied nor the direction changed.
- Gives problems in rocks that are in poor condition.
- Requires highly specialized personnel and previous preparations of the work area.

At the moment there are over 300 rigs in operaton around the world, with the following subsystems of Raise Boring: standard, reversible and blind hole raising.

a) *Standard raise boring*

This is the most widely used system and consists of setting up the equipment on the upper of the two levels to be interconnected, or even outside the mine, so that a pilot hole can be drilled down to a previously opened level. Afterwards, the reamer head is attached to the drill string and the raise is drilled upwards to the rig.

b) *Reversible raise boring*

The same operations are carried out as before, with the difference of placing the equipment on the lower level and inverting the pilot hole and raising execution, which are ascending and descending, respectively.

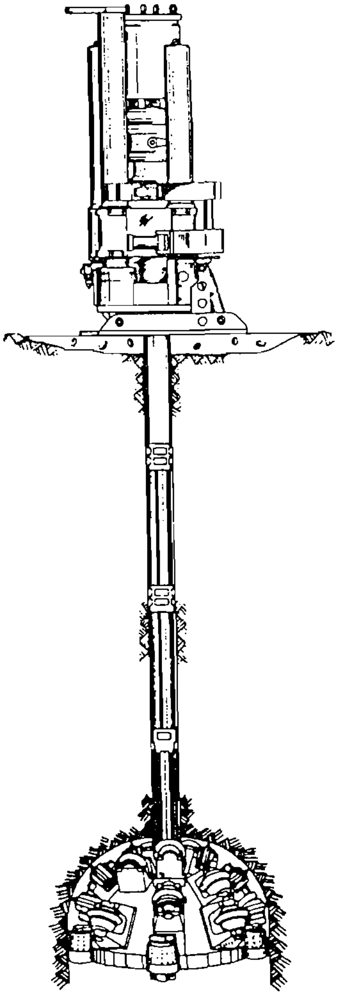


Fig. 7.11. Raising with standard Raise Boring.

c) *Blind hole raise boring*

Once the rig has been erected on the lower level, the drilling is done upwards in full section, without the pilot hole, as there is no access to a second level.

The basic elements to carry out the work, apart from the rig itself which exerts the rotation and feed force from its point of installation are, for the blasthole, the tricone bit, the roller stabilizers and the drill rods; and for the reaming, the axis, base, cutters and their sockets, Fig. 7.12.

The heads can be integral, segmented or extensible. The first are used for diameters from 1 to 3 m with pilot holes of 200 to 250 mm, the segmented for raise diameters that are between 1.5 and 3 m, and the same pilot holes as before, and, lastly, the extensible heads are for sections that range from 2 to 6.3 m with pilot holes up to 350 m.

The power for the equipment is usually over 600 kW with rotary speed, rotary torque and thrust loads on the rock having values that oscillate between: 15 and 30 r.p.m., 150 and 820 kNm and 4 and 12.5 MN, respectively.

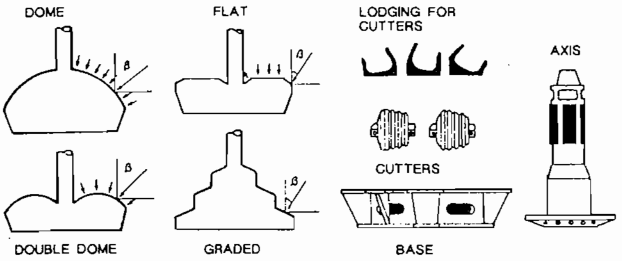


Fig. 7.12. Components of reaming equipment.

7.5 JET PIERCING

This method originated in 1927, when Stores tried to apply it in Germany, in a mine with quartz veins. During the thirties experiments were carried out in taconite beds near Mesabi, and it was after 1947, when with the use of specially designed burners, an efficient drilling was obtained with high output. This was based on the spalling of rock instead of its fusion, thanks to the rapid changes in temperature produced by steam from the cooling water and the combustion gases, which also serve to bail out the chips.

At present time this method has lost field application when compared to large rotary rigs, and it is now only used when cutting ornamental stone.

7.5.1 *Thermal drilling process*

This penetration process depends on one of the rock characteristics which is called spallability and is based on the differential thermal expansion capacity of the rock crystals.

The properties that affect spallability in rocks are complex but the following relationship can be established:

$$\text{Spallability} \propto \frac{\text{Thermal diffusivity} \times \text{Expansion (at } T_0) \times \text{Grain size}}{\text{Compressive strength (at } T_0)}$$

where T_0 is the critical temperature at which the rock becomes plastic enough to prevent spalling.

According to the previous equation, the rocks will be more easily drilled with this method when:

- High thermal expansion exists below 700°C.
- There is high thermal diffusivity below 400°C.
- Granular interlocking structure with no fine clay, mica, or other alteration products.
- Reduced percentage of soft, low temperature melting minerals, or those which decompose.

Some rocks that are apt to spall are: taconites, quartzites, granites, rhyolites, hard sandstones and diabases. In general, the higher the quartz content, the more a rock is apt to spall, as apart from having high coefficients of linear and volume expansion, it has a crystallization change at 573°C, Fig. 7.13.

Rocks with over 30% quartz will spall, as well as those in which a certain amount of water exists in their composition.

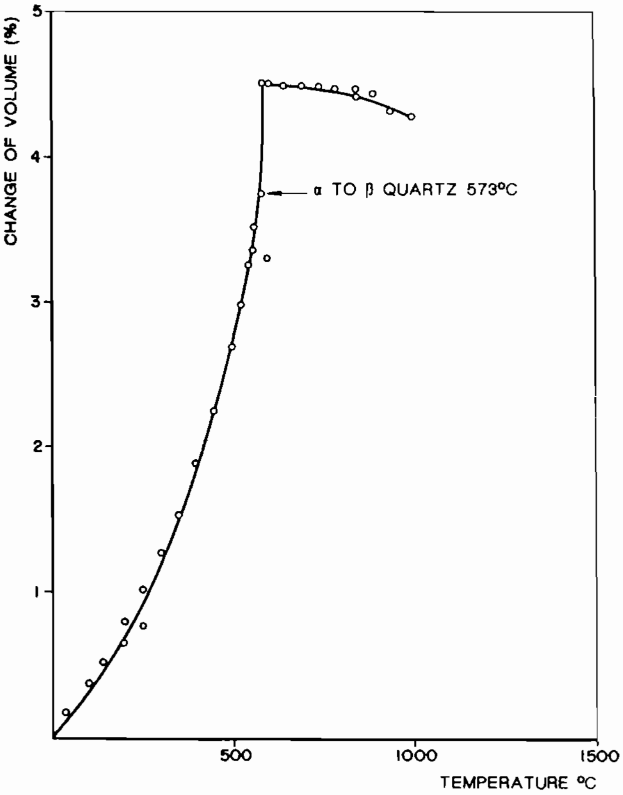


Fig. 7.13. Thermal volume expansion of quartz (Dane, 1942).

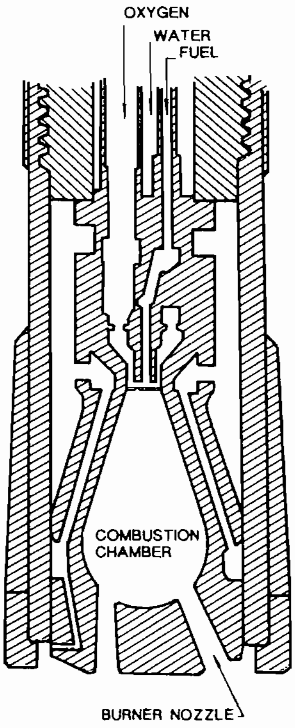


Fig. 7.14. Cross section of a burner.

The basic equipment, or burner, consists of a combustion chamber, Fig. 7.14, where the fuel (gas oil) is atomized and mixed with oxygen as it is maintained under pressure. The injector increases the combustion gas expansion. The flame temperature can reach the end of the burner at 3.000°C when oxygen is injected at 2.000°C if it is compressed air.

The cooling water around the burner avoids its fusion

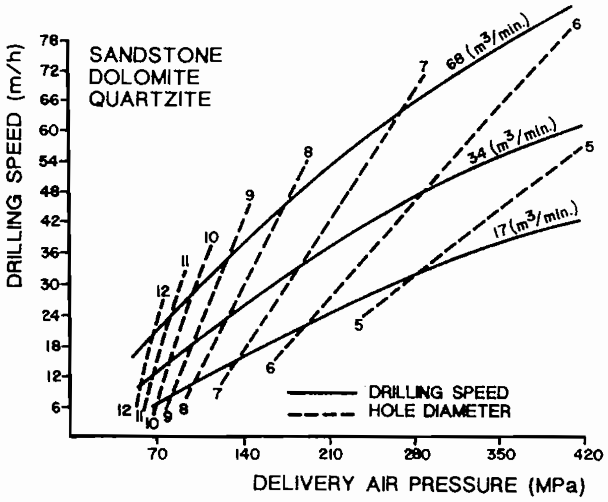


Fig. 7.15. Drilling speeds with compressed air fuel-oil burners.

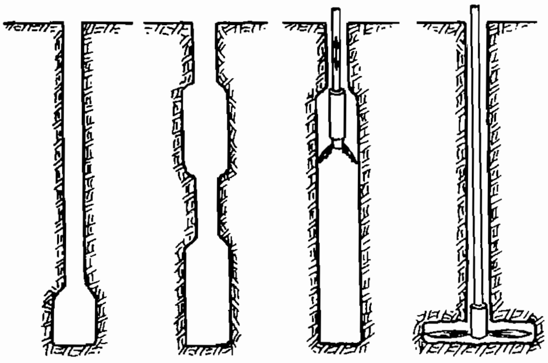


Fig. 7.16. Blasthole enlargement.

and helps in its escape as steam to increase the gases and the bailing out pressure.

With oxygen no special pressure is needed, but it would be necessary with compressed air that is used at 0.7 MPa. In Fig. 7.15, the average penetration rates are indicated, in function with the air flow, pressure and blasthole diameter.

The normal speeds oscillate between 3 and 12 m/h, sometimes reaching 20 m/h under favorable conditions.

7.5.2 *Applications*

The most important applications for this method are:

Blasthole enlargement
This procedure has the following advantages:

- Lower volume of rock drilled per unit broken.
- The explosive column configuration is better as it is closer to l/D = 20, generating higher stresses. The powder factor is lower for a given fragmentation.
- Toe level breakage is improved, reducing subdrilling.
- The volume of stemming is reduced and the confinement of the gases is more effective, also reducing the time taken in said operation.
- The muckpile profile is more adequate for rope shovels.

Stone cutting
Jet piercing is used in ormental granite quarries, in the first phase of liberating the blocks from the rock mass, opening transversal channels in the benches with a width of 60 to 80 mm and a depth that can reach 10 m.

The mounting systems, as with rotary percussive rigs, can be of three types; handheld, and on towable or self-moving frame.

The advantages of thermal drilling are:

- Very hard and abrasive formations can be drilled.
- Blastholes can be enlarged.
- Partial elimination of standard rock breakage by explosive in ornamental stones.
- High drilling speeds in rocks that spall easily.

On the other hand, the following disadvantages exist:

- The machinery is expensive, when compared to large rotary rigs.
- Energy cost is high.
- High noise level and little dust control.

The percentile distribution of costs, according to 'Surface Mining' is as follows:

– Oxygen	31.0%
– Labor	14.4%
– Patent concession	11.0%
– Gas-oil	10.0%
– Reamers	10.3%
– Energy	1.3%
– Maintenance	18.0%
– Water	4.0%

7.6 WATER-JET DRILLING

This technology has gone through spectacular development over the past decade, coupled with the updating of hydraulic equipment of sufficient power, that is robust and trustworthy. At the moment, the system is used in mining to cut ornamental rock and in drilling blastholes for bolting in diameters of 24 and 32 mm.

The equipment basically consists of a high pressure pump, a presure intensifier comprising a piston of double effect and alternative movement capable of carrying out 60 to 80 cycles per minute, and a suitable nozzle. The intensifying effect is achieved by the relative difference of the active surface of the piston, one of which impulses the water through a nozzle of synthetic sapphire with an opening of 0.1 to 1 mm in diameter.

Fig. 7.17 reflects the principle of operation of the pressure intensifier.

The rock breakage, with a high pressure water jet, is produced by the impact and the microcracks that are created in consequence. At a speed of 300 m/s, the pressure created is around 150 MPa, close to the compressive strength of many rocks. With 500 m/s, values of 300 MPa are reached, which is above the compressive strength of the majority of rock materials.

The operative data compiled from equipment being tested are the following:
When drilling blastholes, in order to increase the action of the water jets, tungsten carbide bits are available to carry

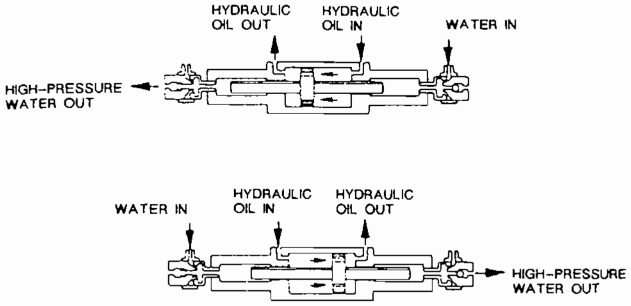


Fig. 7.17. Pressure intensifier equipment.

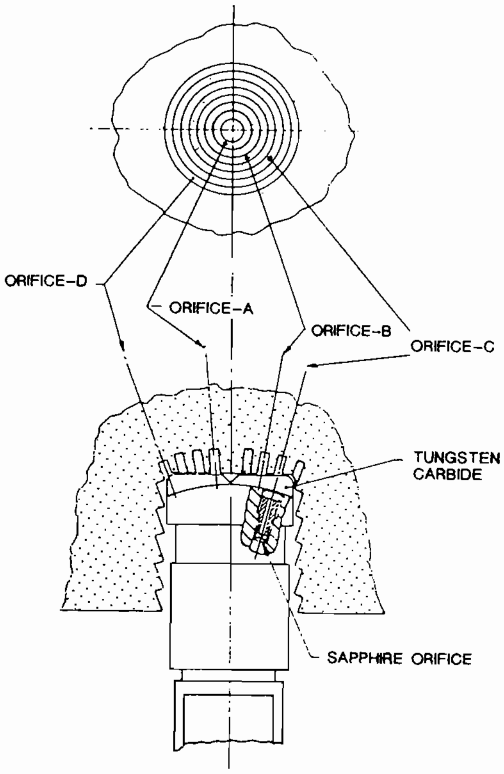


Fig. 7.18. Drill bit with four orifices.

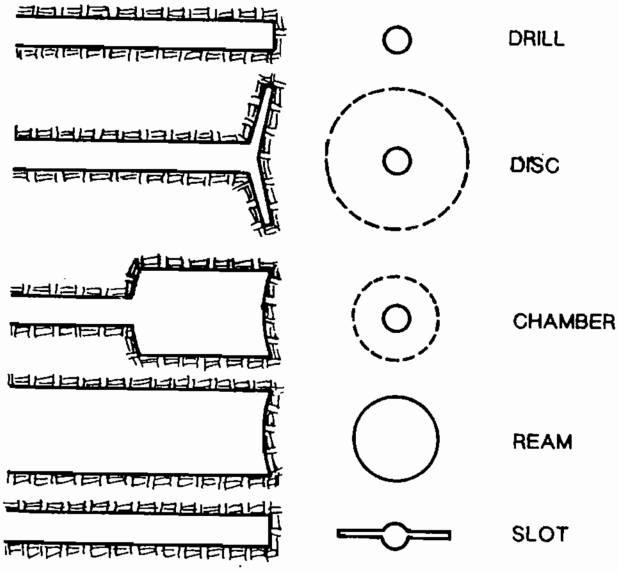


Fig. 7.19. Modification of blasthole geometry drilled with water jet.

Table 7.3.

Intensification ratio	Working pressure (MPa)	Working flow (l/min)
4:1	0-83	19-57
13:1	0-275	5.5-23
20:1	0-378	3.8-15

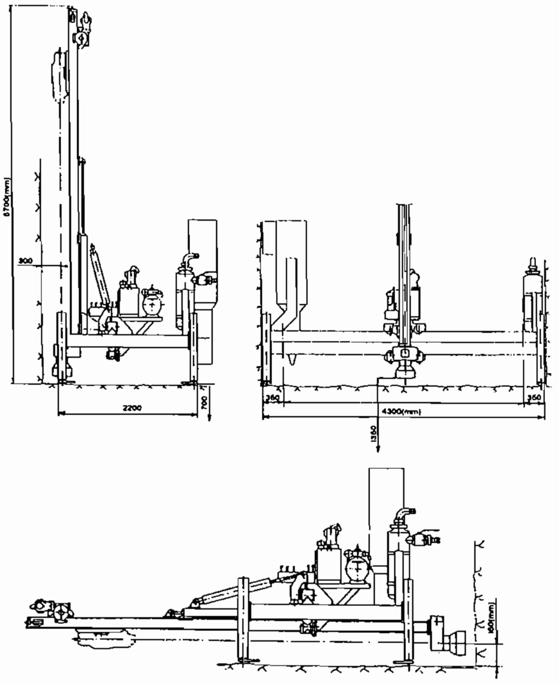


Fig. 7.20. Drilling equipment for dimensional stone.

out the reaming of the concentric ridges of rock that are produced between the grooves cut by the water jets in the bottom of the blasthole, Fig. 7.18.

The application of this breakage technique with explosives opens new horizons, as the blasthole geometry can be modified, thus permitting higher charge concentrations or increases of breaking stresses in certain points of the rock masses.

7.7 DRILLING ORNAMENTAL ROCK

In operations for ornamental rock, such as granite, sometimes special systems are used, as much in primary drilling, where the objective is the separation of a large block from the rock mass, as in the following operations of subdivision and squaring.

Usually hydraulic drills are used, mounted on feeds that move on rails of 3.5 to 4.5 m in length. These themselves can be supported by metal frames mounted on four stabilizing legs or jacks that are directly placed on the ground or on mobile units such as hydraulic shovels, crawlers or tractors on wheels.

The drilling yields are, in the case of the opening of grooves or a lateral channel called *slot drill*, with aligned drying blastholes, of 1.4 to 2.2 m^2/h, and in the primary and secondary drilling with aligned blastholes of 200 to 400 ml/h, according to the power and characteristics of the equipment used.

REFERENCES

Alimak: *Vein Mining*. 1977.

Alimak: *Raise Climber*.

Atlas Copco: *El Método OD*. 1970.

Atlas Copco: *Raise Driving with Jora Lift*. 1970.

Atlas Copco: *Underwater Blasting with the OD Method*. 1971.

Atlas Copco: *ODEX*. 1975.

Atlas Copco: *Rock Drilling Manual – Rock Drilling Tool Applications* 1985.

Calaman, J.J. & Rolseth, M.C.: *Jet Piercing*. Surface Mining. AIME, New York, 1968.

Mohanty, B. & McFarlane, P.: *Water-Jet Drilling and its Applications in Mining and Blasting*. Trans. of the Institution of Mining and Metallurgy, April, 1986.

Pla, F. et al.: *Curso de Perforación y Voladuras*. Fundación Gómez-Pardo. 1978.

Roberts, A.: *Applied Geotechnology*. Pergamon Press. 1981.

Stenmark, E.: *Equipos y Métodos Alimak pare Minería y Obras Subterráneas*. Alimak. 1982.

Trirumalai, K.: *Thermal Drilling of Blastholes*. Mining Engineering Handbook. SME, 1973.

CHAPTER 8

Compressors

8.1 INTRODUCTION

Compressed air is the fluid that has been used as a source of energy in rock drilling, as much for driving the pneumatic equipment with top hammer or down-the-hole hammer, as for flushing of cuttings when drilling with hydraulic or rotary hammers.

For any project, whether it is a surface or underground operation, compressors are a must.

When deciding to purchase a drilling rig, one of the most important aspects is the compressor selection, because of:

- The specific weight on the purchase price which can be, depending on the type of rock drill, between 15 and 55%.
- Its influence on the cost per lineal meter drilled is considerable because if the air flow is insufficient, the following problems can arise:
- Lower penetration rate.
- Increase in costs due to wear: bits, drill rods, etc.
- Increase in fuel consumption.
- More compressor unit maintenance required.

When in the large drilling rigs a high pressure compressor unit is chosen, it will be possible to drill with down-the-hole hammer or with a rolling cone bit.

The two basic features of a compressor, apart from the type or model, are:

- The flow of air supplied.
- The outgoing air pressure.

In Table 8.1, the most frequent values of the mentioned characteristics, the type of compressor and the approximate price percentage in relation to the whole machine, are indicated for different drilling rigs.

8.2 TYPES OF COMPRESSORS

There are two groups of compressors: dynamic and positive displacement. In the first type, the pressure increase is achieved by means of acceleration of the air with an element of rotation and the posterior action of a diffuser. The centrifugal and axial compressors pertain to this group, which is the most adequate for large flows and low pressures.

In positive displacement compressors, which are those that are used in drilling equipment, high pressure is obtained by confining the gas in a closed space, reducing the volume by moving one or various elements. Depending upon the design, they are subdivided into rotary or alternating. The most commonly used in drilling are: piston compressors (reciprocating), when they are stationary, and sliding-vane or rotary screw (helical) for the portable models, either mounted on the drilling unit or towed by it.

8.2.1 *Piston compressors*

These are the best known and oldest as they have been used in underground mines to supply compressed air through the distribution network. Their application has noticeably decreased as a consequence of the massive use of other power sources such as electric and hydraulic.

8.2.2 *Rotary screw (helical)*

In these units the air pressure is achieved through the interaction of two intermeshed helicoidal lobes on the rotors (screws), one male with four lobes and another female with six flutes or grooves. The functioning principle can be observed in Fig. 8.1.

The air enters through the intake port, an inlet formed by the two lobes of the screws and the housing. As the screws rotate the air is trapped and the space between the lobes and flutes reduces, pushing the air farther into the screw, decreasing the volume and therefore compressing the air. Oil is injected to seal the compression chamber, lower its temperature and as a lubricant. As the space occupied by the air and the oil becomes smaller, the oil is gradually emptied into a separator.

This separation is carried out first of all by force of gravity, dropping to the bottom of the receiver tank and then filtered through glass wool. The oil cools, is filtered back to the pump inlet and then recirculates. In Fig. 8.2, the cooling circuits are shown for a portable compressor and its motor.

There are usually two stages in high pressure rotary-screw compressors.

The injected oil has three important missions:

- Sealing the vanes and screws.
- Cooling the air during compression and
- Lubricating the rotors.

The advantages of using Rotary-Screw compressors are:

- They take up little space which makes them ideal for mounting on drilling rigs.
- The set-up is economical.

Table 8.1.

Type of rock drill	Flow (m^3/min)	Pressure (MPa)	Type of compressor	Function operate	% of value of equipment
Pneumatic top hammer	18-36	0.7-0.8	Screw	The hammer, advance motor, translation engine and hydraulic enging.	40-60
Hydraulic top hammer	5-9	0.7-0.8	Screw	Flushing	15-20
Pneumatic down the hole hammer	8-30	0.7-1.75	Screw	Operate the hammer, flushing	40-50
Rotary with tricone bit	18-51	0.3-1.1	Sliding vane (low pressure) screw (medium and high pressure)		10-15

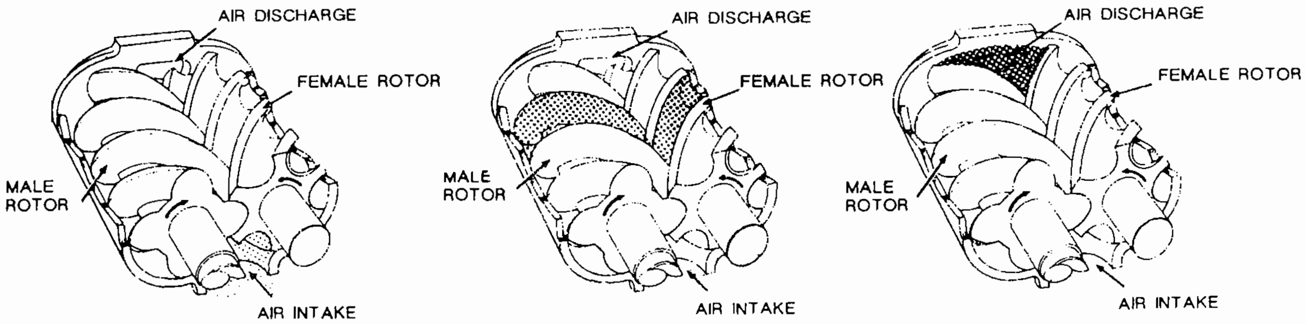


Fig. 8.1. Functioning principle of a helical-lobe (rotary screw) compressor.

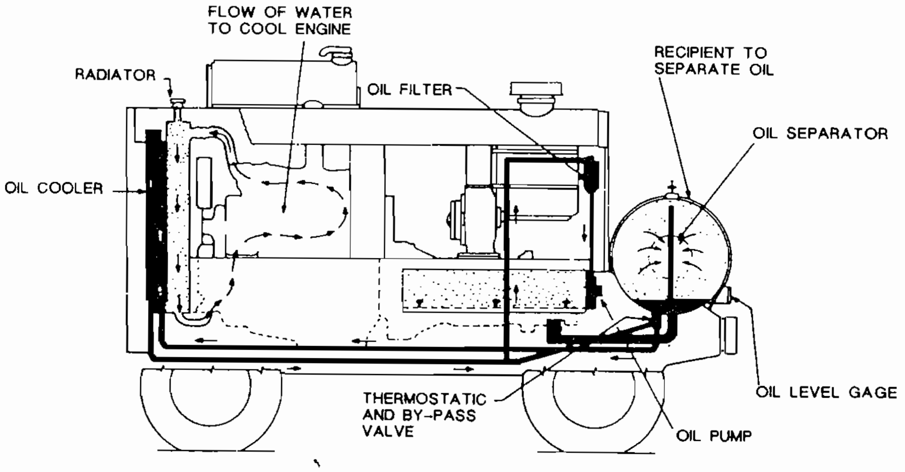

FLOW OF WATER TO COOL ENGINE
RADIATOR
OIL FILTER
RECIPIENT TO SEPARATE OIL
OIL SEPARATOR
OIL COOLER
THERMOSTATIC AND BY-PASS VALVE
OIL PUMP
OIL LEVEL GAGE

Fig. 8.2. Portable compressor (Ingersoll-Rand).

- They produce no important impacts or vibrations.
- Maintenance is minimum.
- Low working temperature and
- High efficiency.

8.2.3 *The Sliding-Vane compressor*

These compressors have only one rotor in cylindrical form or casting, which has longitudinal slots around its circumference that hold sliding vanes or strips. The casting rotates about a longitudinal axis within a housing that is eccentric about the casting. When rotating, the vanes are displaced against the stator by centrifugal force. The air intake, as there are no valves, is aspirated through a port in the housing over which the vanes pass as the casting rotates, trapping the air in the pockets between every two vanes. As the casting rotates the volume diminishes, increasing the air pressure up to the discharge port.

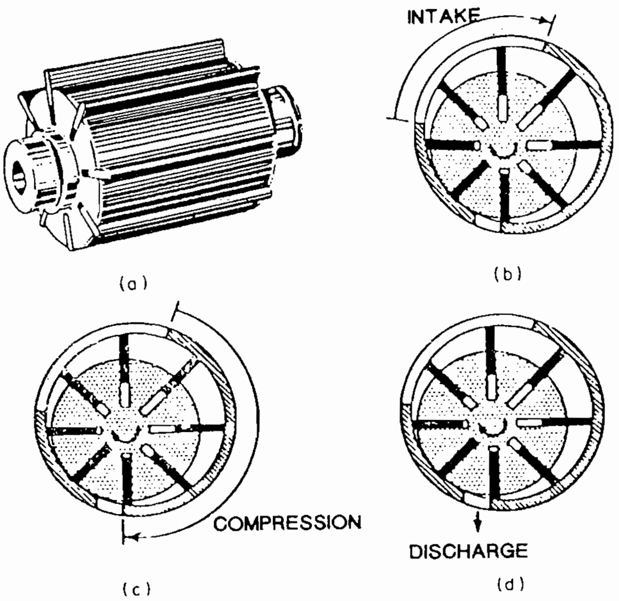


Fig. 8.3. Sliding-Vane rotary compressor.

This type of compressor also uses oil injection for the same reasons as explained previously.

8.3 DRIVE

The stationary compressors are generally driven by electric engines, whereas the portable models, if towed, are run by diesel engines and, if they are mounted on the rigs, by diesel or electric.

In order to compensate a tension drop in the electric engines, there ought to be a surplus power of 10 to 15%.

The connection of the engines to the compressor is carried out by special clamp, trapezoidal belt, direct connection or through a gear train.

8.4 AUXILIARY ELEMENTS

The most important auxiliary elements when working with compressed air are:

- Vacuum filters
- Moisture separators
- Air distribution deposits or banks
- Lubricators
- Pressure multipliers

8.4.1 *Vacuum filters*

To eliminate premature wear on the mobile parts of the compressors and possible break-downs, the air must be filtered before admission. The filters should comply with the following: efficient separation, accumulation capacity, low resistance to air passage, be rugged and easy to maintain.

8.4.2 *Moisture separators*

This element uses the effect of the centrifugal force that the air flow acquires in its rotary movement. The particles make impact on the walls of the collector, therefore drying the air that is evacuated through the central part, Fig. 8.4.

8.4.3 *Air deposits or banks*

Compressed air installations may have regulating deposits or distribution centers. Their size depends upon:

- Compressor capacity
- Regulating systems
- Working pressure
- Estimated variations in air consumption

The functions of the deposits are:

- Storage of compressed air to cover the demands that exceed the compressor capacity.
- Increase cooling and collect the remaining water and oil.
- Equalize the pressure variations in the network.

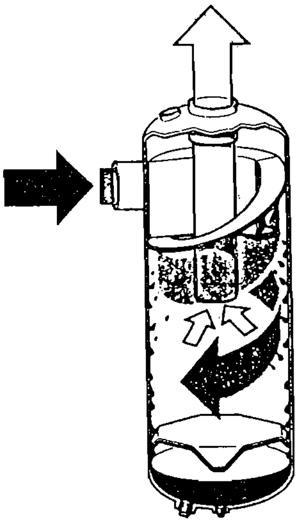


Fig. 8.4. Moisture separator (Atlas Copco).

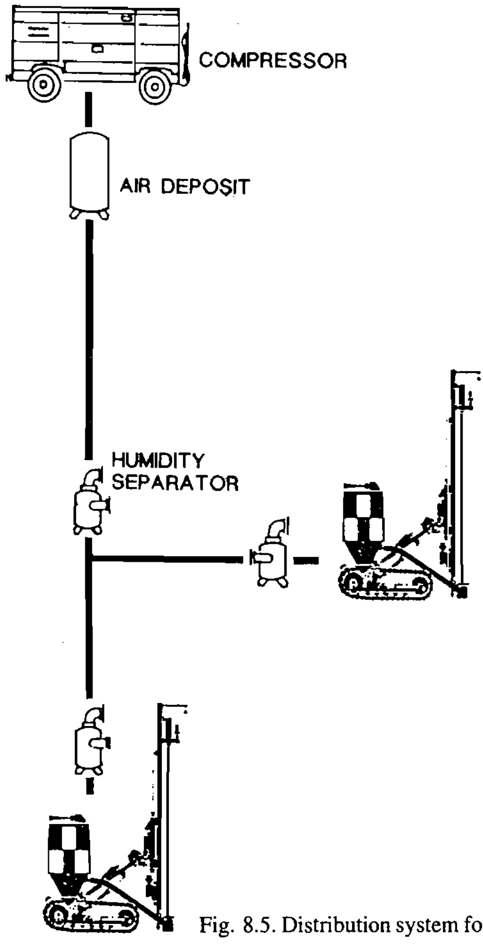


Fig. 8.5. Distribution system for compressed air.

8.4.4 *Lubricators*

To lubricate the rock drills, oil must be added to the compressed air, which can be done in the machine itself or through the air line.

The work principle of the lubricators can be observed

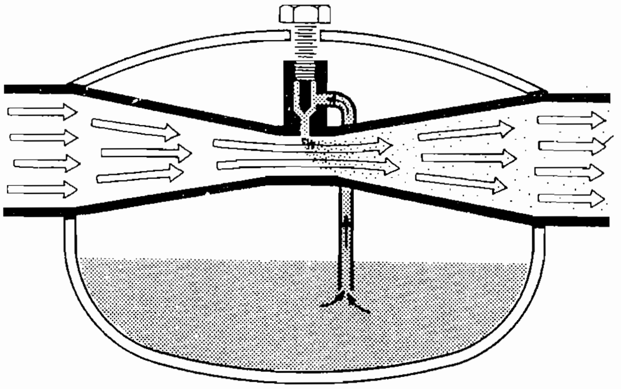


Fig. 8.6. Cross-section of a lubricator.

in Fig. 8.6. the air passes through a strangulator which has a regulating valve. The entering air pressure is connected to the oil tank so that, when the air goes through the narrowest section, its speed increases producing a drop in pressure that makes the oil enter into the air line and become atomized.

8.4.5 *Pressure multipliers (boosters)*

When down-the-hole hammers are used in underground mining, it might be necessary to elevate the air pressure to 1.7 MPa, if it is supplied at half-pressure (0.7 MPa) through stationary installations or when there have been leakages.

The increase in pressure is obtained with *boosters* which work in one or two stages.

8.4.6 *Flexible hoses*

The rubber hoses have outer textile reinforcements placed diagonally, making them flexible and highly resistant. The maximum working pressure is usually 1 MPa, with admissible temperatures that range between –40° to +100°C.

The standard rubber hose sizes that are most commonly used are indicated in Table 8.2.

There are also low weight hoses, one-third the normal weight, manufactured with an inner layer of synthetic fiber inserted in cork that is oil and ozone resistant. It is stored rolled up and flat, which facilitates handling and minimizes storage space. In Table 8.3 some characteristic features of these special hoses are shown.

Table 8.2.

Inner Diameter		Outer diameter		Weight
(mm)	(inches)	(mm)	(inches)	(kg/m)
6.3	¼	12.7	0.50	0.15
10.0	⅜	16.4	0.65	0.19
12.5	½	22.5	0.89	0.35
16.0	⅝	26.0	1.02	0.43
20.0	¾	30.0	1.18	0.54
25.0	1	35.0	1.38	0.78
31.0	1¼	43.5	1.71	0.95
40.0	1½	52.0	2.05	1.15
50.0	2	66.0	2.60	1.80
63.0	2½	79.0	3.11	2.20
80.0	3	96.0	3.78	2.50
100.0	4	116.0	4.57	4.20

Table 8.3.

Inner diameter weight		Outer diameter				Max. working pressure
(mm)	(inches)	(mm)	(inches)	(mm)	(psi)	(kg/m)
20	¾	24	0.9	1.2	175	0.160
25	1	30	1.2	1.2	175	0.230
40	1½	45	1.6	1.2	175	0.390
50	2	56	2.3	1.2	175	0.550
76	3	82	3.3	0.8	116	0.850

A whole group of elements are used for the connections, from threaded and claw couplings, connectors, clamps, etc.

8.5 CALCULATING PRESSURE DROPS

In stationary installations, an acceptable drop in pressure between the compressor and the farthest point of consumption is around 10 kPa.

In very long distribution lines, and especially in temporary work areas, the costs of the installations are usually decisive. In such cases, the pressure drop should not exceed 50 kPa.

In order to estimate the pressure drops in compressed air distribution systems, the nomograph of Fig. 8.7 can be used.

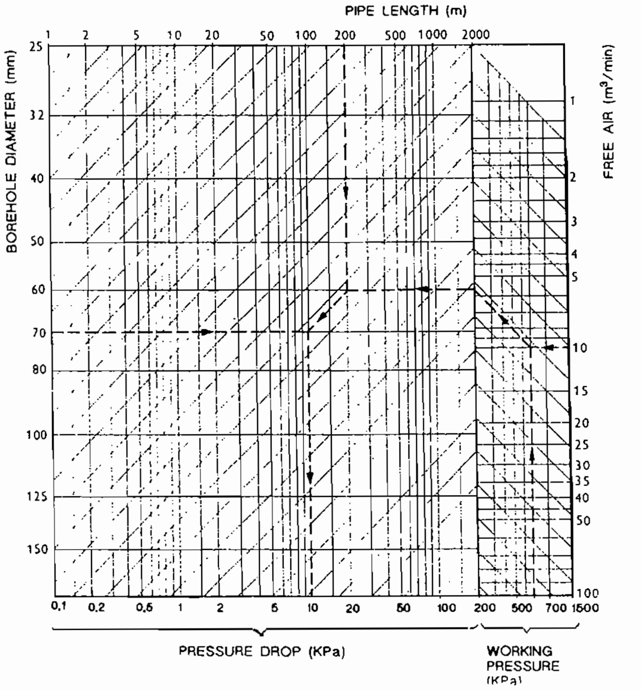


Fig. 8.7. Nomograph to calculate the sizes of the compressed air conducts and pressure drops.

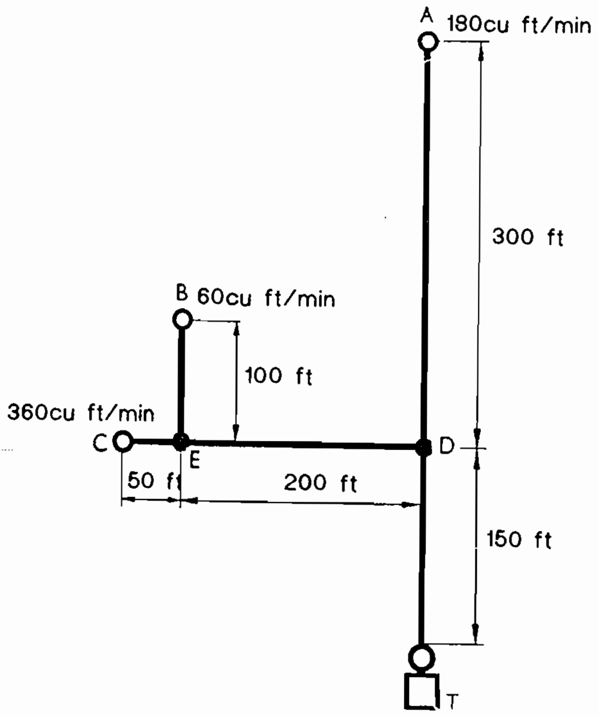


Fig. 8.8. Pattern of the compressed air network.

Example 1
The pressure drop is to be calculated in a compressed air installation that consists of a 200 m hose with an inner diameter of 70 mm. The initial air pressure is 800 kPa and and the flow of 170 l/s.

By following the pattern of darker lines drawn in the nomograph, a pressure drop of 10 kPa is obtained.

Example 2
In Fig. 8.8, the pattern of a compressed air network is shown, in which at the discharge port of the compressor *T*, the air branches out to the different consumer points A, B, and C, where the flows consumed are respectively of 5, 1.5 and 10 m^3/min. The maximum admissible pressure drop between the compressor and the consumer points is set at 10 kPa. Without taking into consideration the pressure drops in other points such as narrow spaces, bends, etc., the proper sizes for the different conducts is to be found for the indicated lengths.

Using the nomograph in Fig. 8.7, the following is obtained:

Section T-D. Pipe length l = 50 m, Air flow Q = 16.5 m^3/min, Working pressure P = 0.7 MPa.
A tentative inner diameter for the tube is 80 mm, for which the pressure drop is 3.5 kPa.

Section D-A. l = 100 m, Q = 5 m^3/min, P = 0.7 MPa.
The pressure drop should not go over 10 kPa less the pressure drop of section T-D, which is to say, 10 – 3.5 kPa = 6.5 kPa.

For an inner diameter of 50 mm, the pressure drop is of 6.5 kPa, and the global drop between T and A will then be 10 kPa.

Section D-E. l = 60 m, Q = 11.5 m^3/min, P = 0.7 MPa.
The inner diameter of the tube is chosen for a maximum pressure drop of 5 kPa. For a diameter of 70 mm, the pressure drop would be 3.8 kPa.

Section E-B. l = 30 m, Q = 1.5 m^3/min, P = 0.7 MPa.
The drop in pressure should not be more than 10 kPa less the pressure drop between T and E, which is to say 10 – (3.5 + 3.8) = 2.7 kPa. The diameter should be 32 mm, which gives a pressure drop of 2.4 kPa. The global drop will be 9.7 kPa.

Section E-C l = 15 m, Q = 10 m^3/min, P = 0.7 MPa.
The pressure drop should not go over 2.7 kPa. The best inner diameter is 60 mm, which produces a drop in pressure of 1.5 kPa and a total drop between T and C of 7.7 kPa.

REFERENCES

Atlas Copco: *Manual Atlas Copco.* Cuarta Edición, 1984.
Ingersoll Rand: *Compresores Portátiles Rotativos.* 1976.
Rundquist, W.A.: *Know Your Compressor.* Pit & Quarry. September, November, 1978; February, 1979.

CHAPTER 9

Thermochemistry of explosives and the detonation process

9.1 INTRODUCTION

The essential objective in the use of explosives for rock breakage consists in having a chemically concentrated energy, properly placed and in sufficient quantity so that when it is liberated in a controlled manner, in time and space, it can achieve the fragmentation of rock material.

The explosion is, according to Berthelot, 'the sudden expansion of gases in a volume much larger than the initial, accompanied by noise and violent mechanical effects'.

The types of explosion are the following: mechanical, electric, nuclear and chemical. From the point of view of this handbook, only the last are of interest.

Commercial explosives are those that are a mixture of compounds, some combustible and some oxidizing which, when properly initiated, have an almost instantaneous exothermic reaction that generates a series of high temperature gaseous products that are chemically more stable and take up a larger volume.

To be able to get an idea of the strength of an explosive, a comparison with other sources of energy can be carried out. Let us consider for a moment a thermal plant with 550 MW of power installed. Knowing that 1 kW is equal to 0.238 kcal/s, the power installed is the equivalent of 130.900 kcal/s.

One kilogram of gelatin explosive is 1.000 kcal/kg placed in a column of 1 meter in length and with a detonation velocity of 4.000 m/s releases a strength of:

$$\frac{1200 \text{ kcal}}{1 \text{ m}/4000 \text{ m/s}} = 48 \times 10^5 \text{ kcal/s}$$

which is 37 times higher than that of the thermal plant.

The time factor of the explosion is so highly important that even when the commercial explosives have a small calorie strength (TNT, 1.120 kcal/kg) compared with other combustible compounds (Anthracite, for example, 7.000 kcal/kg), the speed of reaction of an explosive is such that when it detonates on a metal plate it can make a hole, because there is no time for the developed energy to dissipate towards the edges or upwards where it is opposed by the resistance of air.

The gases produced accumulate the heat generated, dilating to a volume that can be around 10.000 times more than that of the blasthole where the explosive is located.

In this chapter the detonation mechanism of explosives will be analyzed as well as some basic concepts of thermochemistry.

9.2 DEFLAGRATION AND DETONATION

Chemical explosives, depending upon the conditions to which they are exposed, can offer different behavior than would be expected from their explosive nature. The decomposition processes of an explosive compound are: the actual combustion, the deflagration and, lastly, the detonation. Both the nature of the compound itself, as well as the initiation system and the external conditions govern the decomposition process.

a) *Combustion*
This can be defined as any chemical reaction capable of giving off heat, whether it is actually felt by our senses or not.

b) *Deflagration*
This is an exothermic process in which the transmission of the decomposition reaction is mainly based upon thermal conductivity. It is a superficial phenomenon in which the deflagration front advances through the explosive in parallel layers at a low speed which, usually, is not over 1.000 m/s.

c) *Detonation*
This is a physicochemical process characterized by its high speed of reaction and the formation of large quantities of gaseous products at an elevated temperature, which build up a great expansive force. In the detonating explosives, the speed of the first gasified molecules is so great that they do not lose their heat through conductivity to the unreacted zone of the charge but transmit it by shock, deforming it and provoking its heating and adiabatic explosion, generating new gases. The process repeats itself with ondulating movements that affect the whole explosive mass and is called shock wave.

The initiation energy can be supplied in various ways, depending upon the explosive in question. In the low explosives (deflagrating) or powderss, the energy of a flame is sufficient; whereas in high or detonating explosives a shock wave type of energy is necessary.

Once the explosive has been initiated, the first effect produced is the generation of a shock or pressure wave that propagates through its own mass. This wave is the carrier of the energy necessary to activate the molecules of the explosive mass that is around the initial energized focus which then starts a chain reaction.

At the same time that this wave is produced, the

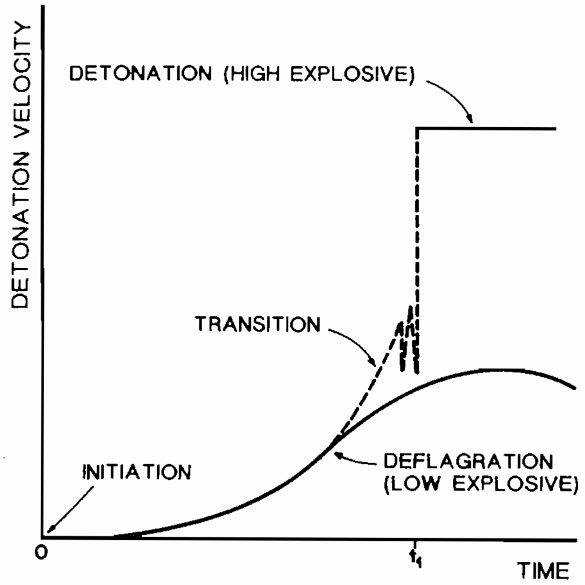


Fig. 9.1. Detonation process.

reacting explosive mass releases a large quantity of high temperature gases. If this secondary pressure acts upon the rest of the mass that is undetonated, its effect is added to that of the primary pressure wave, passing from the process of deflagration to that of detonation, Fig. 9.1.

If the gas pressure wave performs to the contrary in the unreacted explosive mass, a system of slow deflagration takes place, slowing down the explosive reaction and causing a loss of energy in the primary detonation wave that can even become incapable of energizing the rest of the explosive mass, thereby stopping detonation.

9.3 DETONATION PROCESS OF AN EXPLOSIVE

As described before, the detonation consists in the propagation of a chemical reaction that moves through the explosive at supersonic speed, transforming it into new chemical compounds. The basic characteristic of these reactions is that it is initiated and sustained by a supersonic shock wave.

As indicated in Fig. 9.2, an initial shock starts the chemical transformation that takes place through the reaction zone to end in the so called Chapman-Jouguet (C-J) plane where chemical balance is produced, at least in ideal detonations.

In commercial explosives, important chemical reactions are developed behind the C-J plane, especially reactions of the ingredients in large particle form and the combustible metals. These secondary reactions can affect the performance of the explosive but do not have influence upon the stability or velocity of detonation.

In a high strength explosive the primary reaction zone is usually quite narrow, only millimeters, while in low density and strength explosives this zone is much wider and, in the case of ANFO, can be of several centimeters.

The products of the reaction are behind the C-J plane, and in some instances so are the inert particles. The majority of the products are gases which reach a temperature of 1.500 to 4.000°C and pressures that go from 2 to 10 GPa. The gases under these conditions expand rapidly and produce a shock or strain wave in the surrounding media.

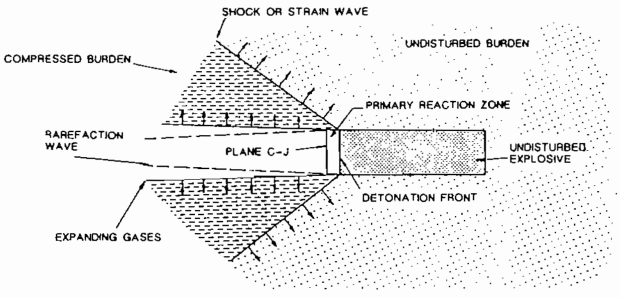


Fig. 9.2. Detonation process of an explosive charge.

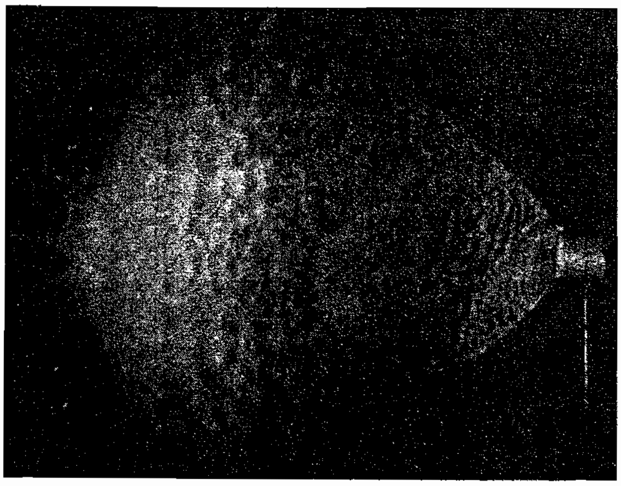


Photo 9.1. Detonation of an unconfined charge (Nitro Nobel).

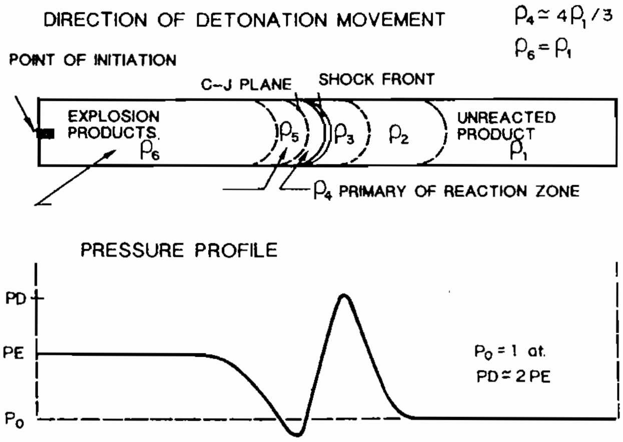


Fig. 9.3. Pressure profile in the detonation of an explosive column.

In Fig. 9.3 a simplified profile of an explosive column is shown. The detonation wave, which is characterized by a sudden rise in pressure, displaces itself towards the right at supersonic speed. Behind the maximum pressure level a contraction is produced as a consequence of momentary conservation that compensates the forward impulse by generating a rarefaction wave that is transmitted in the opposite direction.

The C-J plane moves at very high speed *VD*, whereas the speed of movement of the explosion products, determined by Cook with x-ray photographs, reaches a value of 0.25 detonation velocity. For this reason, if the maximum pressure of the explosive wave is:

$$PD = \rho_e \times VD \times U_p$$

where: PD = Detonation pressure, ρ_e = Density of the explosive, VD = Detonaton velocity, U_p = Particle velocity.

And taking ito account that $U_p = 0.25 \times VD$, the following is obtained:

$$PD = \frac{\rho_e \times VD^2}{4}$$

where: PD = Detonation pressure (kPa), ρ_e = Density of the explosive (g/cm^3), VD = Detonation velocity (m/s).

The thermochemical pressure or maximum pressure available to carry out an operation *PE*, is considered to be worth half the detonation pressure. If the explosive charge is well in contact with the rock wall of the blasthole, the pressure exerted upon it by the explosion gases is the same as the thermochemical pressure.

With reference to the initiation of the explosives, it is necessary to apply a certain amount of energy per volume unit at a given spot. One of the theories used to explain the initiation mechanism is called of the *hot spots*, which are small elements of matter in which the energy supplied globally to the explosive is found. The hot spots can be formed by the adiabatic compression of small air, gas or steam bubbles that have been retained in the explosive, by friction between the crystals that make up the explosive compound, or by heating produced in the viscous movement of the explosive mass under extreme conditions.

When the hot spots receive a certain amount of energy, the surrounding explosive mass decomposes, producing a release of energy that then can create more hot spots, starting in this manner a chain reaction. This is the basis of sensitivation for some blasting agents, by means of adding crystal microspheres, solid particles, etc.

9.4 THERMOCHEMISTRY OF THE EXPLOSIVES

The thermochemistry of explosives refers to the changes of internal energy, mainly in form of heat. The energy stored in an explosive is potential energy, latent or static. The potential energy which is released by the detonation process is transformed into kinetic or mechanical energy.

The law of energy conservation establishes that in any isolated system the total amount of energy is constant, although it may be transformed:

Potential Energy + Kinetic Energy = Constant.

However, not all the energy supplied is useful, as some loss always takes place.

There are two alternative methods that can be used to calculate the changes in energy: one, the application of the known laws of physics and chemistry, and the other by analysis of the final products. The latter is more complex, as the products that can be conveniently analyzed are rarely present at the moment of maximum pressure and temperature. Because of this, it is frequently necessary to make theoretical calculations based upon knowledge of the laws of chemistry and physics in order to predict the properties of explosives or the parameters of detonation.

An approximate calculation of these parameters can be made for explosives that have zero balance oxygen, or close to it, as with these in the ideal explosion only CO_2, H_20, N_2, and 0_2 are produced and it is possible to apply the thermodynamic method of analysis. When the explosives are not oxygen balanced, it becomes necessary to use an iterative system of non-lineal equations to find the detonation parameters.

In the following, the most important thermochemical parameters are explained along with the simplified method of calculation.

9.5 HEAT OF EXPLOSION

When there is an explosion at constant pressure, with the only work done being that of expansion or compression, the first law of thermodynamics establishes that:

$$Q_e = -\Delta (U_e + P \cdot V)$$

where: Q_e = Heat released by the explosion, U_e = Internal energy of the explosive, P = Pressure, V = Volume.

As $U_e + P \cdot V$ refers to heat content or enthalpy H_p, then $Q_e = -\Delta H_p$ can be written. Thus, the heat of explosion at constant pressure is equivalent to the change in enthalpy and can be estimated by establishing the thermal balance of the reaction, multiplying the heats of end product formation by the number of moles formed by each, and adding them, to later subtract the heats of formation of the explosive.

$$\Delta H_{p\,(\text{explosive})} = H_{p\,(\text{products})} - H_{p\,(\text{explosive})}$$

If ANFO is taken into consideration, for example, then:

Table 9.1. Heats of formation and molecular weights of some explosives and explosion products (Konya & Walter, 1990).

Compound	Formula	Mol weight	Heat of formation (kcal/mol)
Corundun	Al_2O_3	102.0	–399.1
Fuel oil	CH_2	14.0	– 7.0
Nitromethane	CH_3O_2N	61.0	– 21.3
Nitroglycerine	$C_3H_5O_9N_3$	227.1	– 82.7
PETN	$C_3H_8O_{12}N_4$	316.1	–123.0
TNT	$C_7H_5O_6N_3$	227.1	– 13.0
Carbon Monoxide	CO	28.0	– 26.4
Carbon Dioxide	CO_2	44.0	– 94.1
Water	H_2O	18.0	– 57.8
Ammonium Nitrate	NH_4NO_3	80.1	– 87.3
Alumnium	Al	27.0	0.0
Carbon	C	12.0	0.0
Nitrogen	N	14.0	0.0
Nitrogen Oxide	NO	30.0	+ 21.6
Nitrogen Dioxide	NO_2	46.0	+ 8.1

$$3NH_40_3 + 1CH_2 \rightarrow C0_2 + 7H_20 + 3N_2$$

$$H_{p\,(\text{explosive})} = 3\,(-87.3) + (-7) = -268.0\text{ kcal}$$

$$H_{p\,(\text{products})} = (-94.1) + 7\,(-57.8) + 3(0)$$
$$= -498.7\text{ kcal}$$

$$Q_{mp} = \Delta H_{p\,(\text{explosive})}$$
$$= -[-498.7 + 268.9] = 229.8\text{ kcal}$$

As the molecular weight of the explosive P_m is:

$$P_m = 3\,(80.1) + (14) = 254.3\text{ g}$$

The Heat of Explosion per resulting kilogram is:

$$Q_{kp} = \frac{229.8\text{ kcal}}{254.3\text{ g}} \times 1000\text{ g/kg} = 903.7\text{ kcal/kg}$$

The heat released at constant pressure has no technical interest because the detonation process takes place at constant volume. Therefore, to calculate the latter, it will be necessary to increase the heat at constant pressure with the work consumed in the adiabatic expansion.

$$Q_{mv} = Q_{mp} + 0.58 \times n_{pg}$$

where: n_{pg} = Number of moles of the gaseous products.

And if instead of the heat released per mole, that corresponding to a kilogram of explosive is wished to be known, then:

$$Q_{kv} = \frac{Q_{mv} \times 1000}{P_m}$$

Thus, the following will result from the previous example:

$$Q_{mv} = 229.8 + 11 \times 0.58 = 236.18\text{ kcal/mole}$$

$$Q_{kv} = \frac{236.18 \times 1000}{254.3} = 928.74\text{ kcal/kg}$$

If there are solid products among those of the explosion, SiO_2, Al_2O_3, chlorides, carbonates, etc., heat from the reaction is used in their fusion during the first phase of the explosion. Because of this, the total heat calculated must be diminished with that which corresponds to the product of the solid component by the latent heat of fusion.

9.6 OXYGEN BALANCE

With the exception of nitroglycerine and AN, the majority of explosives are oxygen-deficient, because they do not contain enough oxygen to convert each atom of carbon and hydrogen present in the explosive molecule into carbon dioxide and water. Normally, an explosive does not utilize atmosphere oxygen during the detonation process. Thus, the heat generated by explosion of an oxygen-deficient explosive is less than that generated under conditions of complete oxidation.

The oxygen balance is expressed as a final percentage that is equal to the difference between 100% and the calculated percentage (oxygen present ÷ oxygen required). In the case of oxygen deficiency, the final percentage is given a negative sign. In many explosives, the sensitivity, strength and breaking power increase with increasing oxygen balance, reaching a maximum at the zero balance point.

Therefore, in TNT the following exists:

$$2\,CH_3C_6H_2\,(NO_2)_3 \rightarrow 12\,CO + 2CH_4 + H_2 + 3N_2$$

It can be seen that 16.5 moles of O_2 are required to reach zero oxygen balance of 2 moles of TNT, or 8.25 moles of O_2 per mole of TNT. The oxygen balance for this reaction will be:

$$100\% - \left(\frac{3.00}{8.25} \times 100\right) = 63.6\%,\ \text{expressed as} - 63.6\%$$

In explosives that are oxygen-positive, the available oxygen will combine with the atoms of carbon to form CO_2 and nitrogen oxides, some of which are red in color. Red fumes indicate insufficient fuel in the reaction which can be due to poor mixing, segregation or a loss of fuel.

The oxygen-negative explosive form incomplete oxides, CO in particular, which is poisonous and colorless. Nitrous fumes are much reduced, and for this reason many explosives are formulated with a small negative oxygen balance.

9.7 VOLUME OF EXPLOSION

This is the volume that is occupied by the gases of one kilo of explosive under normal conditions. The molecular volume, or volume of 1 gram-mole of any gas under normal conditions, is 22.4 liters.

If one considers NG as an example, the following exists:

$$4\,C_3H_5\,(NO_3) \rightarrow 12\,CO_2 + 10H_2O + 6N_2 + O_2$$

The explosion of 1 g-mole of NG generates 29/4 = 7.25 g-moles of gaseous products at 0°C and at atmosphere pressure meaning that the explosion volume will be:

$$7.25\text{ g-mole} \times 22.4\text{ l/g-mole} = 162.4\text{ l}$$

At a higher temperature, the volume of gases increases according to the law of Gay-Lussac. Therefore, in the previous example, an increase of 15°C would give:

$$162.4 \times \frac{283}{273} = 171.3\text{ l}$$

Normally, the volume of explosion is expressed in terms of moles of gas per kilogram of explosive.

$$\frac{n_{pg} \times 1000}{n_{ex} \times P_m}$$

where: n_{pg} = Moles of gaseous products, n_{ex} = Moles of explosive, P_m = Molecular weight of explosive.

For the previous example, a value of 31.9 moles of gas per kg of NG is obtained.

9.8 MINIMUM ENERGY AVAILABLE

Minimum energy available is the quantity of work that can be done by the gaseous products of an explosion when the pressure remains constant at 1 atm. For example, when nitroglycerine detonates, the molecular volume increases more than 700 %, while the resisting pressure remains constant.

The differential equation for the work of expansion w_e is:

$$dw_e = F_e \times dl$$

where: F_e = Magnitude of the force, dl = Element of distance through which the force is applied.

As the force is equal to the pressure per unit of surface area, therefore:

$$dw_e = P \times A_s \times dl$$

but as $A_s \times dl$ is the volume change of the gaseous products, as P is constant, the following exists:

$$\int_{w_{e1}}^{w_{e2}} dw_e = P \times \int_{V_1}^{V_2} dV$$

or

$$w_e = P \times (V_2 - V_1)$$

where: w_e = Work of expansion, P = Pressure (resisting 1 atm), V_1 = Volume of explosive, V_2 = Volume of the gaseous products of explosion.

As the volume of the explosive V_1 is negligible against the volume of gases produced V_2, the amount of work available is given by:

$$w_e = P \times V_2$$

In the case of 1 g-mole of nitroglycerine, substitution in the previous equation gives:

$$w_e = 1 \text{ atm} \times 171.3 \text{ liters} = 171.3 \text{ liters} = 171.3 \text{ liters-atm}$$

$$w_e = 1 \times 171.3 \times 10.23 = 1.752.4 \text{ kgm}$$

This quantity of work is considered the minimum energy available.

9.9 TEMPERATURE OF THE EXPLOSION

As in any combustion, the absolute temperature is given by:

$$T_e = \frac{Q_{kv}}{\Sigma(m_r \times c_e)}$$

where: Q_{kv} = Total heat released at a constant volume, m_r = Weight in Kg of each of the products of the reaction, c_e = Specific heats at the temperature T_e.

As $c_e = f(T_e)$, Kast and Beyling published some functions of $c_e = a - b / T_e$ for each of the products, from which the following can be established:

$$m_1 \times a_1 - \frac{m_1 \times b_1}{T_e}$$

$$m_2 \times a_2 - \frac{m_2 \times b_2}{T_e}$$

$$m_n \times a_n - \frac{m_n \times b_n}{T_e}$$

from where:

$$T_e \times [m_1 \times a_1 + m_2 \times a_2 \text{-----} + m_n \times a_n - \frac{1}{T_e} (m_1 \times b_1 + m_2 \times b_2 + \text{-----} + m_n \times b_n)] = Q_{kv}$$

thus:

$$T_e = \frac{Q_{kv} + \Sigma m_r \times b}{\Sigma m_r \times a}$$

The function $a - b/T_e$ for the explosion products are:

Of the steam (0.943 – 1153/Te) kcal/kg.
Of nitrogen (0.234 – 49.0/Te) kcal/kg.
Of remaining oxygen (0.212 – 34.4/Te) kcal/kg.
Of carbon oxide (0.246 – 67.7/Te) kcal/kg.
Of carbon dioxide (0.290 – 87.8/Te) kcal/kg.

For the solid products, the influence of the temperature is lower and it can be neglected, therefore taking:

ClNa	0.219 kcal/kg
SiO_2	0.190 kcal/kg
Al_2O_3	0.200 kcal/kg
CO_3Na_2	0.362 kcal/kg
CO_3Ca	0.320 kcal/kg
MnO	0.201 kcal/kg
CO_3K_2	0.278 kcal/kg
ClK	0.172 kcal/kg
CaO	0.228 kcal/kg
Fe_2O_3	0.145 kcal/kg

9.10 PRESSURE OF THE EXPLOSION

For perfect gases, the relationship $P \times V = R \times T_e$ is fulfilled, but for the real gases, the Law of Van der Waals can be applied:

$$\left(P + \frac{a}{V^2}\right) \times (V - b) = R \times T_e$$

In explosives, the value $V - b/V^2$ is neglected, from which the so called Law a Sarrau ($b = \alpha$) is deducted

$$P \times (V - \alpha) = R \times T_e$$

The value of α would be the equivalent of zero for perfect gases, in which $V = 0$ when $P = \infty$. When this does not occur, $V = \alpha$ which represents the volume of the gaseous molecules.

If there is also an incompressable solid body when $P = \infty$, this solid will retain its volume α′. Later the *Covolume*, in the gases and products of an explosion, is equal to α + α′. Although this is a theoretical concept, it is used for approximate calculations, making α equal to the volume of the gases at 20°C per kilogram of explosive divided by 1.000, that is to say:

$$\alpha = \frac{V_{k20}}{1000}$$

maintaining α' with a specific value.

If in Sarrau's equation V (volume of the blasthole equal to that of cartridge, approximately, in the first phase of the explosion) is taken into consideration and a density of the explosive ρ_e, expressed in kg/l, for one kilo of explosive, then

$$V = \frac{1}{\rho_e}$$

and therefore:

$$P = R \times T_e \times \frac{\rho_e}{1 - \alpha \times \rho_e}$$

which is known as the equation of Noble and Abel. The product $R \times T_e$ is called Specific Pressure or Force $- f_s$, which can be interpreted as the pressure of 1 kg of explosive that will occupy one liter of volume and whose explosion gases were perfect $\alpha = 0$. It is a theoretical concept that simplifies some formulas and allows the comparison of explosives.

From f_s, the value of P is obtained.

$$P = \frac{f_s}{V - \alpha} \text{ and as } \frac{1}{V} = \rho_e$$

the pressure P in MPa, when ρ_e is expressed in kg/m^3, is given by:

$$P = f_s \times \frac{\rho_e \times 10^{-1}}{1 - \alpha \times \rho_e}$$

or

$$P = 0.03526 \times V_{K20} \times R_e \times \frac{\rho_e}{1 - \alpha \times \rho_e}$$

The value of α has been estimated by Hino (1959) from the specific volume V_s (volume of the explosive or of the blasthole between the explosive mass):

$$\alpha = 0.92 \times [1 - 1.07 \times e^{-1.39 \times V_s}]$$

REFERENCES

Aguilar, F.: *Los Explosivos y sus Aplicaciones*. Editorial Labor, S.A. 1978.

Konya, C.J. & E.J. Walter: *Surface Blast Design*. Prentice Hall, 1990.

Manon, J.J.: *The Chemistry and Physics of Explosives*. E/MJ. January, 1977.

Muñiz, E.: *Apuntes de Explosivos*. E.T.S. Ingenieros de Minas de Madrid. 1968.

Preston, C.J.: *New Methods in Production Blast Monitoring and OptimizatioN* Du Pont Canada, Inc. 1983.

CHAPTER 10

Properties of explosives

10.1 INTRODUCTION

Conventional explosives and blasting agents have different properties which characterize them and are used in assessing them for correct selection, depending upon the type of blasting to be carried out and the conditions under which it will be effectuated.

The properties of each group of explosives also give prediction of the probable results of fragmentation, displacement and vibrations.

The most important characteristics are: strength and energy developed, detonation velocity, density, detonation pressure, water resistance and sensitivity. Other properties which affect their use and must be taken into account are: fumes, resistance to high and low temperatures, desensitivation by external causes, etc.

10.2 STRENGTH AND ENERGY

The strength is, from the industrial application point of view, one of the most important properties, as it defines the energy available to produce mechanical effects.

There are different terms to express the strength of an explosive. In the original dynamites (straight dynamite), the percentage of nitroglycerin was the parameter to measure the strength. Later, with the partial substitution of nitroglycerin with other products and the carrying out of comparative laboratory tests, the terms were changed to Relative Weight Strength and Relative Bulk Strength. This way, it is frequent to refer to the strength of an explosive in so much percent of another taken as a standard, such as pure gelatin dynamite, ANFO, etc., which has the assigned value of 100.

There are various practical methods to measure the strength or the available energy of an explosive, all disputable due to their peculiarities and their repercussion on the results when compared to the yields obtained in the blasts.

10.2.1 *Traulz test*

This method determines the expansion capacity that the detonation of 10 g of explosive produces in the center of a solid lead block, Fig. 10.1. The difference between the initial volume of 62 cm^3 and the total obtained volume gives the true Traulz value.

When the volume is compared to that produced with 7 g of picric acid, the Traulz Index is obtained. If the reference explosive is gelatin dynamite, the strength will be expressed as percentage in relation to the same.

As the explosives with most strength tend to give an increase in volume larger than would correspond to their real strength, the CERCHAR defined the Coefficient of Practical Utilization 'C.U.P.', which is based upon the comparison of explosive weights C_{ex} which produce volumes that are equal to that of a standard charge of 10 or 15 g of picric acid.

$$\text{C.U.P.} = \frac{15}{C_{ex}} \times 100$$

10.2.2 *Ballistic mortar test*

This consists in comparing the propulsion of a steel mortar mounted upon a ballistic pendulum by the effect of the gases when a charge of 10 g of explosive is detonated. The T.B.M. index is calculated from the equation:

$$\text{T.B.M.} = 100 \times \frac{1 - \cos\alpha}{1 - \cos\beta}$$

where α and β are the angles registered in the recoil deflection of the pendulum corresponding to the test explosive and the standard explosive.

The two procedures described give good results with dynamite-type explosives but are not applicable in blasting agents such as ANFO or watergels because of:

– The small diameter used in the pendulum, 20 mm, and in the Traulz test, 25 mm, as they are smaller than the critical diameter of those explosives.

– The 2 cm of stemming used in the Traulz block is projected by the gases before they can perform effectively.

– The charge is frequently decoupled in the mortar.

– And, above all, these tests are only adequate when the explosives are cap-sensitive and the reaction times are small.

10.2.3 *Seismic Strength test*

This method consists in detonating an explosive charge in an isotropous rock medium, and register the seismic disturbance produced at a determined distance.

The standard explosive is usually ANFO and it is

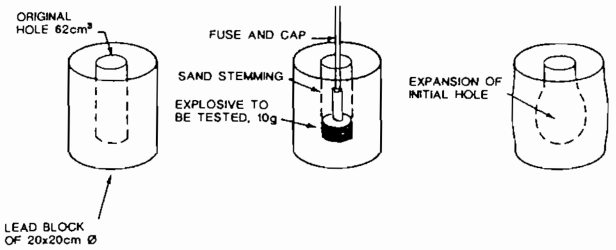


Fig. 10.1. Traulz block.

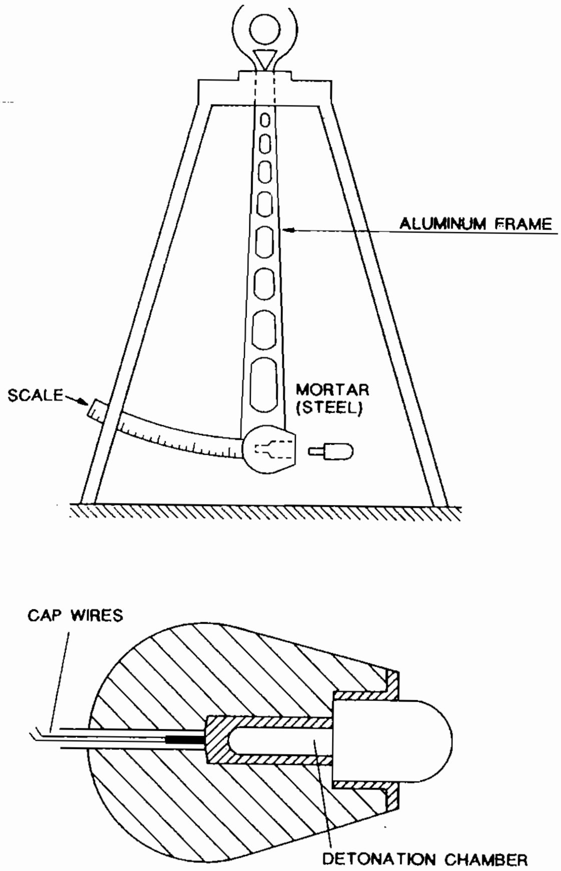


Fig. 10.2. Ballistic mortar test.

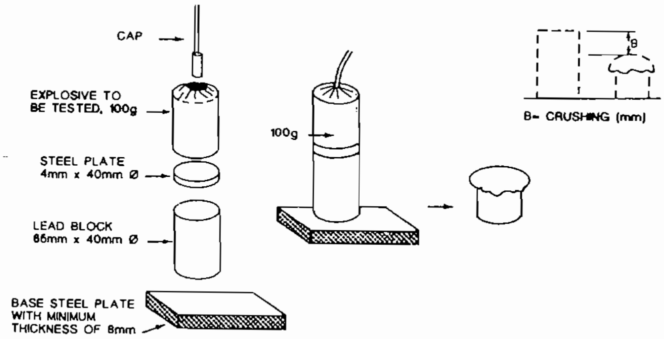

CAP
EXPLOSIVE TO BE TESTED, 100g
STEEL PLATE 4mm x 40mm Ø
LEAD BLOCK 65mm x 40mm Ø
BASE STEEL PLATE WITH MINIMUM THICKNESS OF 8mm
100g
B
B- CRUSHING (mm)

Fig. 10.3. Hess test.

supposed that the variation in vibrations is proportional to the energy of the explosive raised to 2/3. This test is not considered very adequate in measuring the available energy of an explosive.

10.2.4 *Crater charges test*

This is based upon the selection of the Critical Depth and the Optimum Depth, which are those in which an explosive charge breaks the surface rock and produces the largest volume crater, respectively.

The main problem of this system is the difficulty in having an available test area of homogeneous rock and in the number of shots necessary.

10.2.5 *Cylinder compression test*

This method defines the Breaking Power of an explosive, which is related to the fragmentation capacity of the rock, by means of the squashing produced by a charge upon a lead cylinder. There are various methods such as that of Kast and of Hess, but the latter is the most common, Fig. 10.3.

This test gives a good idea of the strain energy which is linked to the detonation pressure.

10.2.6 *Plate dent method*

A cylindrical charge is detonated upon a steel or aluminum plate. The dent formed in the plate gives a quantitative measurement of the energy of the detonation. The results of this test are subject to wide variations unless the geometry of the explosive charge, point and system of initiation are maintained identical, and are favorably biased towards explosives with a high strain wave energy, Photo 10.1.

10.2.7 *Double pipe test*

A steel pipe is charged with the explosives to be tested, and is placed parallel to and in contact with a second steel witness pipe of the same length. Upon detonation, the witness pipe is squashed, and the indentation bears a relationship to the energy generated by the test explosives, Fig. 10.4.

This test method can provide a limited degree of continuous recording of explosive performance over the full

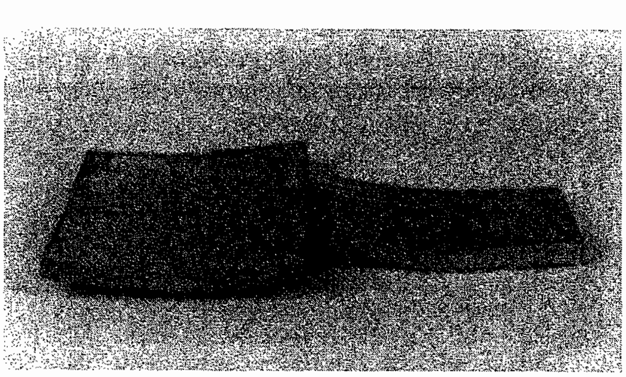


Photo 10.1. Test on plate.

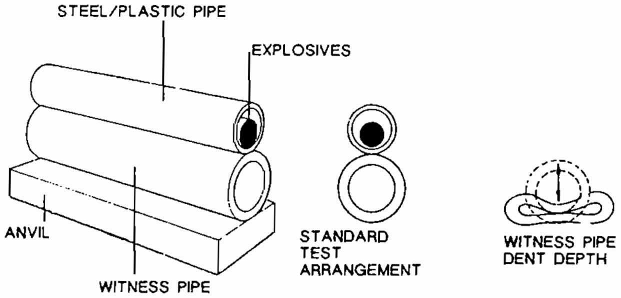


Fig. 10.4. The Double Pipe Test.

length of the charge. It can give information on detonation *run-up* zones and fade-out zones, and indicate the effect of discontinuities along the charge length. Alternatives to the pipe are a witness plate of steel or aluminum, or a slab of lead, known as the Traulz Block.

10.2.8 *Underwater test*

This technique to quantify energy released by an explosive was suggested by Cole almost 30 years ago, and it is characterized as being one of the most complete as it permits tests with charge geometries similar to those in blastholes. Another feature is that it permits the separate calculation of the energy linked to the strain wave, which we shall call from now on Strain Energy-ET, and the energy of the detonation gases, called Bubble Energy-EB, as well as the possibility of evaluating the influence of the initiation system on the energy released by an explosive, Fig. 10.5.

According to Blanc (1984) if $p_h(t)$ is the pressure of the hydraulic shock wave and t_B is the first pseudoperiod of oscillation of the bubble formed by the gases after detonation, the following is obtained:

$$\mathrm{ET} = \frac{4\pi DS^2}{\rho_0 \times VH} \times \int_{t_1}^{t_2} p_h(t) \times \mathrm{dt}$$

$$\mathrm{EB} = K \times \left[\frac{P_h^5}{\rho_0^3}\right]^{1/2} \times t_B^3 \quad \text{(Willis formula)}$$

where: DS = Distance of the charge from the pressure captator, ρ_0 = Volumetric mass of the water, VH = Speed of the shock wave in the water, t_1, t_2 = Integration interval, K = Constant, P_h = Total pressure at the location of the submerged charge (hydrostatic + atmospheric).

This method is very useful for comparing the yields of similar explosives under the same testing conditions. At the moment, it is the most used procedure in evaluating explosive energy, because with exception of the Thermic Energy, the rest is reliably assessed.

10.2.9 *Empirical equations*

1. The Swedish equation proposed to evaluate Relative Weight Strength PRP of an explosive is:

$$\mathrm{PRP} = \frac{5}{6} \times \frac{Q_e}{Q_o} + \frac{1}{6} \times \frac{VG}{VG_o}$$

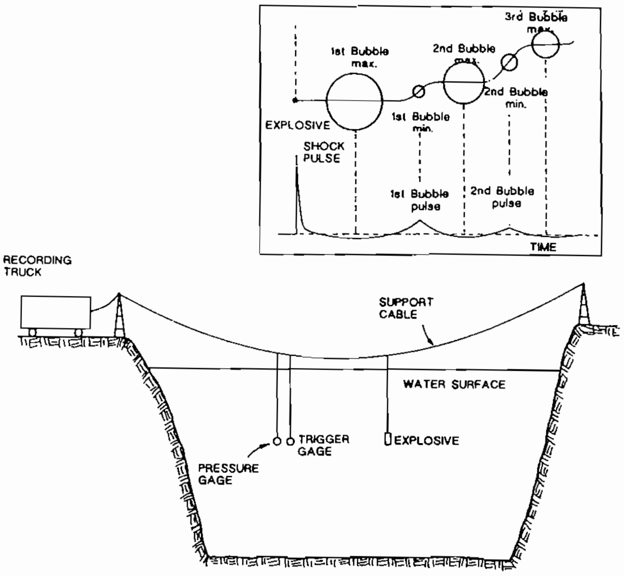


Fig. 10.5. Underwater blastings to assess explosives energy.

where: Q_o = Heat of the explosion of 1 kg of explosive LFB (5 MJ/kg) under normal conditions of pressure and temperature, Q_e = Heat of the explosion of 1 kg of explosive to be used, VG_o = Volume of the gases released by 1 kg of explosive LFB (0.85 $\mathrm{m^3/kg}$). VG = Volume of the gases liberated by the explosive to be used.

As in some occasions the strength refers to ANFO, the strength can first be calculated with respect to the standard explosive LFB and the value obtained can be divided by 0.84 which is the relative strength of ANFO with respect to said explosive. ANFO has values of Q_e and VG of 3.92 MJ/kg and 0.973 $\mathrm{m^3/kg}$ respectively.

2. Paddock (1987) suggests comparing the explosives be means of the so called Strength Factor defined by

$$FP = PAP_x \times VD \times \rho_e$$

where: PAP_x = Absolute Weight Strength of the explosive (x) (cal/g), VD = Detonation velocity (m/s), ρ_e = Density of the explosive ($\mathrm{g/cm^3}$).

If ANFO is taken as the standard explosive, the following will be fulfilled:

$$PAP_{\mathrm{ANFO}} = 890\ \mathrm{cal/g}$$

$$PAV_{\mathrm{ANFO}} = PAP_{\mathrm{ANFO}} \times \rho_e \times 890 \times 0.82 = 730\ \mathrm{cal/cm^3}$$

$$PRP_x = \frac{PAP_x}{PAP_{\mathrm{ANFO}}}$$

3. Another equation used to calculate the Relative Weight Strength is

$$PRP = \left(\frac{\rho_e \times VD^2}{\rho_o \times VD_o^2}\right)^{1/3}$$

where: ρ_e = Density of the explosive ($\mathrm{g/cm^3}$), VD = Detonation velocity (m/s), ρ_o and VD_o refer to the standard explosive.

10.3 DETONATION VELOCITY

Detonation velocity refers to the speed with which the detonation wave is propagated through the explosive and, therefore, is the parameter which defines the rhythm of energy release.

The factors that affect *VD* are: charge density, diameter, confinement, initiation and aging of the explosive. For the first three, as the parameters increase, the resulting *VD* grow significantly, Fig. 10.6.

As to initiation, if it is not sufficiently energetic the detonation could start with a low speed; and if aging is a factor, this makes the *VD* diminish as the number and volume of the air bubbles is lower, especially in gelatin explosives which are generators of hot points.

There are diverse methods to measure *VD*, amoung which the following are emphasized:

- D'Autriche method
- Kodewimetro
- Chronograph

10.3.1 *D'Autriche method*

This method is based upon comparing the *VD* of the explosive with the velocity that is already known of a detonating cord. A cord of a certain length is taken and the exact middle is marked, making it coincide with a mark made on a lead plate upon which it is placed. Afterwards, the ends of the cord are inserted in the explosive at a prefixed distance *d*. The explosive charge, which can be encased in a metal tube, is initiated at one of the ends with a detonator. As the detonation wave then initiates, in different instants, the ends of the cord, the collision of the two detonation waves takes place upon the plate at a distance *a* of the middle point of the cord. Therefore, the VD_e of the explosive is determined from:

$$VD_e = \frac{VD_c \times d}{2a}$$

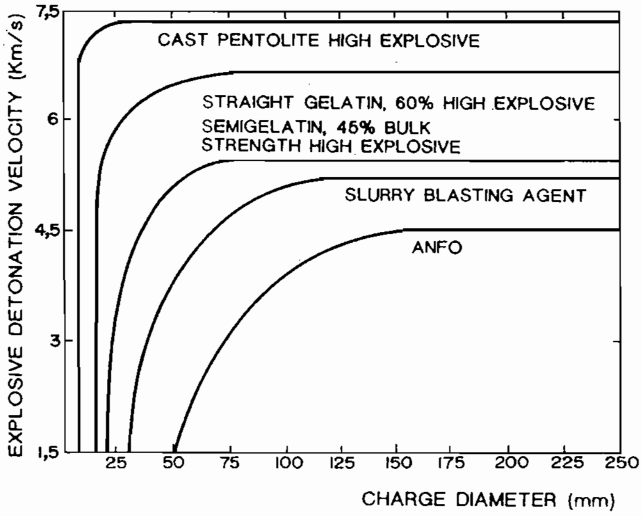


Fig. 10.6. Influence of the charge diameter upon the detonation velocity (Ash, 1977).

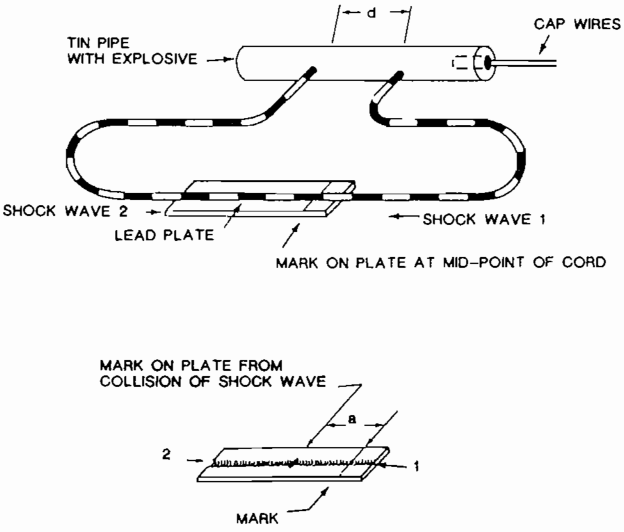


Fig. 10.7. D'Autriche method.

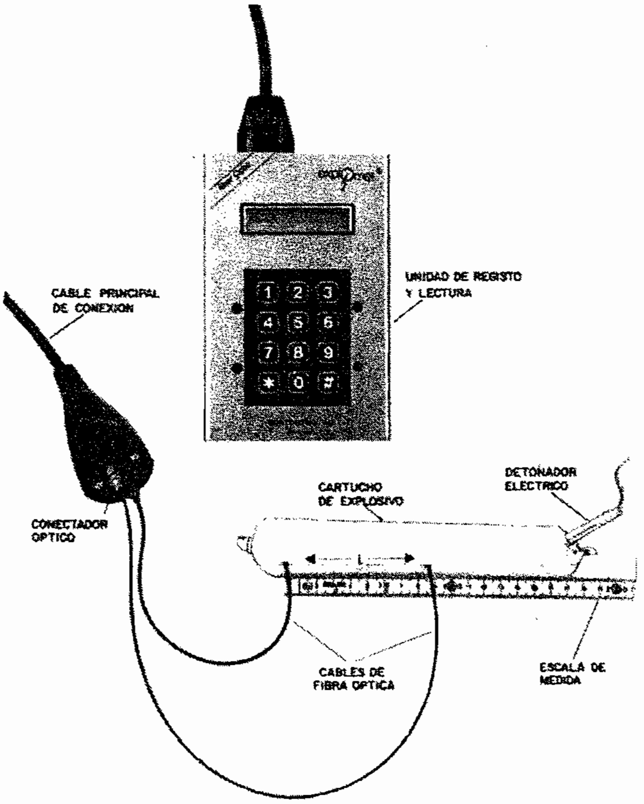


Photo 10.2. Measuring detonation velocity with a chronograph.

10.3.2 *Kodewimetro*

This is based upon the variation in resistance of a cable probe that goes through an explosive column axially. By means of special equipment called Kodewimetro, connected to an oscilloscope, the voltage variation, which is proportional to the resistance, is measured while maintaining a constant current intensity in the circuit. As the detonation wave advances along the length of the explosive, the electric resistance diminishes and the *VD* can be determined from the stress to which it is proportional.

10.3.3 *Chronograph*

From two sensors introduced in the explosive and placed at a determined distance the *VD* can be calculated by measuring the activation time of each sensor. At the moment, there are instruments that are capable of giving *VD* directly and with high precision. The sensors can be electric or, more modernly, of optical fiber, Photo 10.2.

10.4 DENSITY

The density of the majority of explosives varies between 0.8 and 1.6 g/cm^3 and, as in detonation velocity, the greater it is, the more breakage it provides.

In blasting agents, density can be a critical factor because, if it is very low, they become sensitive to the detonating cord which begins to initiate them before the detonation of the primer cartridge; on the other hand, if it is very high they can become insensitive and not detonate. That limit density is called 'Death Density' which will be explained further on.

The density of an explosive is an important factor in calculating the necessary amount of charge for a blast. As a general rule, in the bottom of the blastholes where more energy concentration is required, higher density explosives such as gelatin explosives or watergels are used; whereas in column charges lower density is required and ANFO based or powder explosives are used.

The lineal charge concentration q_l in a blasthole of diameter D and a density ρ_e, is calculated from:

$$q_l \text{ (kg/m)} = 7.854 \times 10^{-4} \times \rho_e \times D^2$$

where: ρ_e = Explosive density (g/cm^3), D = Charge diameter (mm).

10.5 DETONATION PRESSURE

The detonation pressure of an explosive is a function of the density and of the square of the detonation velocity. It is measured in the C-J plane of the detonation wave when it is propagated through the explosive column, as already indicated.

Although the detonation pressure of an explosive depends upon, apart from the density and the *VD*, the ingredients of which it is composed, this parameter can be estimated from the following equation:

$$PD = 432 \times 10^{-6} \times \rho_e \times \frac{VD^2}{1 + 0.8 \times \rho_e}$$

where: PD = Detonation pressure (MPa), ρ_e = Density of explosive (g/cm^3), VD = Detonation velocity (m/s).

The commercial explosives have a *PD* that varies between 500 and 1.500 MPa. Generally, in hard and competent rocks the fragmentation is done more easily with high detonation pressure explosives, owing to the direct relationship that exists between this variable and the breakage mechanisms of the rock.

10.6 STABILITY

Explosives should be chemically stable and not decompose under normal atmospherical conditions. A method to check the stability is called the Abel test, which consists in heating a sample during a set time and at a specific temperature, observing the moment in which decomposition initiates. For example, nitroglycerine at 80° takes 20 minutes to decompose.

The stability of the explosives is one of the properties that is related to the maximum storage time of these substances, so that their effects in a blast will not be reduced.

10.7 WATER RESISTANCE

This is the capacity to resist a prolonged exposure to water without losing its characteristics. It varies with the composition of the explosive and is generally linked to the proportion of nitroglycerine or special additives that they contain; thus watergels, gelatin dynamites and emulsions are quite resistant to water. Oxidizing salts, such as ammonium nitrate in ANFO, intensely diminish their water resistance because they are very hygroscopic.

The scale of classification generally accepted goes from Null, Limited, Good, Very Good and Excellent. In the first, the explosive has no resistance to water, while the last guarantees a resistance of more than 12 hours.

10.8 SENSITIVITY

This characteristic globally envolves various meanings that depend upon the type of external action that affects the explosive.

– *Controlled action.* The sensitivity is equivalent to the acceptance of detonation by an initiator (e.g. electric blasting cap).

– *Uncontrolled action.* The sensitivity is a measure of the ease with which an explosive can be detonated by heat, friction, impact or shock.

Photo 10.3. Water resistance.

10.8.1 *Cap sensitivity*

Explosives should be sufficiently sensitive to detonation by an adequate initiator. This capacity varies depending upon the type of product, therefore, as an example, the majority of gelatin explosives are initiated by electric blasting caps, whereas blasting agents generally require a multiplier or primer of higher pressure and detonation velocity.

The sensitivity to initiation test is carried out on a lead plate on which an explosive cartridge is placed with set dimensions and with different shots the minimum cap strength necessary is established. One of the classifications used is the following: Explosives that are No. 8 cap sensitive and those that are not (non cap sensitive). Said caps, which are the most commonly used, have a charge of a mixture of 2 g of mercury fulminate (80%) and potassic chlorate (20%), or a charge of equivalent pressed pentaerythritotetranitrate (PETN).

10.8.2 *Sensitivity to shock or friction*

Some explosives can detonate by means of subsonic stimulants such as shock or friction. For safety purposes it is important to know their degree of sensitivity when faced by these actions, especially during handling and transportation.

The shock resistance test is usually carried out with a drop hammer (Kast), which consists in placing a sample of the product on an anvil, around 0.1 g, upon which a steel weight of 0.5 to 10 kg is dropped from different heights until explosion is achieved and a drop distance established.

For example, with a hammer of 2 kg, mercury fulminate detonates with a drop distance of 1 to 2 cm, nitroglycerine with 4 to 5 cm, dynamite with 15 to 30 cm, and ammoniacal explosives have drop distances of 40 to 50 cm.

The most common friction test is the Julius Peter test, in which an explosive is subjected to a friction process between two porcelain surfaces that have not been polished and upon which different pressures are exerted. After the test, any carbonization is noticeable as well as deflagration or explosion. The results are expressed in kilos which correspond to the pressure with which the porcelain surface rubs upon the plate where the explosive is deposited.

10.8.3 *Sensitivity to heat*

When explosives are heated gradually, they arrive at a temperature in which they suddenly decompose with release of gases, increasing little by little until finally a deflagration is produced or a small explosion. That temperature is called Ignition Point.

In black powder it varies between 300 and 350°C, and in industrial explosives between 180 and 230°.

This characteristic is different from sensitivity to fire, which indicates inflammable properties. Therefore, black powder, which has a good degree of sensitivity to heat, is very inflamable, exploding with just a spark, as also occurs with nitroglycerine.

10.8.4 *Critical diameter*

Cylindrical shaped explosive charges have diameters below which the detonation wave does not propogate or, if it does, it is with a velocity below the standard rate. This diameter is called 'Critical Diameter'.

The principal factors that influence the critical diameter of an explosive are: particle size, the reactivity of its components, the density and the confinement of said components.

10.9 DETONATION TRANSMISSION

Sympathetic transmission is the phenomenon produced when a cartridge, upon detonation, induces the explosion of another that is adjacent.

A good transmission within the blastholes is the guarantee that the explosive columns will be completely detonated. However, when those blastholes are close, or the charges within are decked, a sympathetic detonation can be produced by transmission of the strain wave through the rock, by the presence of underground water and structural discontinuities, or by pressure of the inert material of intermediate stemming upon the adjacent charges. In all these cases the results of fragmentation and vibrations would be seriously impaired.

One of the methods to measure the capacity or aptitude for sympathetic detonation, also defined as the Coefficient of Autostimulation, consists in calculating the maximum distance at which a primed cartridge can make another non-primed receiver cartridge explode, both being aligned with reference to their axis and well in contact with a ground or metal surface, or even inside tubes of different materials or in the open air.

In the majority of the industrial explosives, the maximum distances at which sympathetic detonation is produced are between 2 and 8 times their diameter, depending upon the type of explosive. The measures of the Autostimulation Coefficient can be taken either directly or indirectly; however, in the second, only 50% of the energy given by the direct method is transmitted.

The factors which modify the results of these tests are: aging, the caliber of the cartridges and the method used in testing.

As to the transmission of detonation between cylindrical charges with inert barriers, there has been little practical investigation, because the majority of these tests

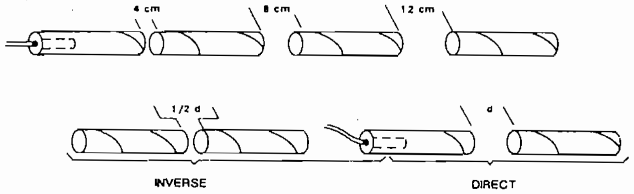


Fig. 10.8. Test of sympathetic detonation.

were carried out by placing homogeneous solid or liquid materials between the primed and non-primed charges instead of the granular materials used in intermediate stemmings such as gravel or drill cuttings.

10.10 DESENSITIZATION

In many industrial explosives it has been observed that the sensitivity diminishes when the density increases beyond a certain value. This phenomenon is more acute in those compositions or blasting agents that do not contain substances such as TNT or Nitroglycerine, etc.

In watergels and ANFO-type mixtures the variation in sensitivity is much greater than in gelatin explosives.

In Fig. 10.9, the influence of the density of ANFO upon the *VD* can be observed. Above the values of 1.1 g/cm^3, the velocity falls drastically, so the densities and pressures which produce those levels of confinement are called 'Death Densities and Pressures'.

The desensitization can be caused by:

- Hydrostatic pressures and
- Dynamic pressures.

The first is usually only present in very deep blastholes, and is not common for this reason.

In dynamic sensitization three situations can be distinguished.

10.10.1 *Desensitization by detonating cord*

Detonating cord of half core load does not initiate watergels and emulsions properly and can even make them insensitive to other priming systems.

The explication for each different type of explosive is not always the same.

– For ANFO, the detonating cord, according to its strength, initiates it partially or creates a weak detonation.

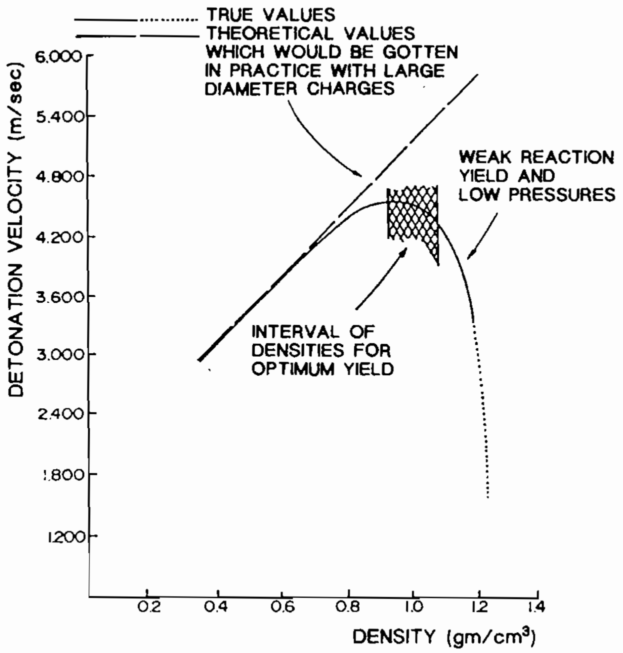


Fig. 10.9. Detonation velocity of ANFO as a function of density.

– In watergels, the cords are insufficient to create a stable detonation wave, compressing the generatrix bubbles of the *hot points*, making them insensitive to the effects of a primer or a posterior shock wave.

– In emulsions, low strength detonating cords can break the forseen structures of the composition, giving the explosive its sensitivity for posterior priming.

All these phenomena depend largely upon the diameter of the charge.

10.10.2 *Desensitivation by channel effect*

If a column of cartridged explosives is placed in a blasthole with a larger diameter, the detonation of the charge is accompanied by a flow of gases that expands through the empty anular space, compressing the air. At high pressure, the air exerts a lateral force upon the explosive, ahead of the detonation front, provoking an increase in density and, in consequence, a desensitivation of the explosive which can cause a drop in the detonation velocity.

10.10.3 *Exerted pressure by adjacent charges*

Desensitization caused by the detonation of adjacent charges can be due to:

– A shock wave that passes through the charge from the adjacent ones.

– Lateral deformation of the blasthole, compressing the charge, due to rock movement or underground water.

– Compression of the charge by pushing of the intermediate stemming material and,

– By infiltration of explosion gases through fissures or fractures opened in the mass.

10.11 RESISTANCE TO LOW TEMPERATURES

When the environmental temperature is under 8°C, the explosives which contain nitroglycerine tend to freeze. By adding a certain amount of nitroglycol, the freezing temperature is lowered to –20°.

10.12 FUMES

The detonation of any commercial explosive produces steam, nitrogen, carbon dioxide, and eventually, solids and liquids. Among the harmless gases mentioned, there is always a certain percentage of toxic gases such a

Table 10.1. Fume classification (Institute of Makers of Explosives, USA)

Class	Volume of toxic gases (CO-NO$_2$)–dm^3
1	0 – 4.53
2	4.53 – 9.34
3	9.34 – 18.96

carbon monoxide and nitrogen oxides. These resulting products are called fumes.

According to the proportion of harmful gases, a scale of classification has been established in degrees of toxicity for the operators after blasting.

These figures refer to the gases produced by the firing of a charge of 200 g of explosive, with its paper wrapping in the so-called 'Bichel Bomb'. Depending upon that classification, the 1st class explosives can be used for any type of underground work, those of 2nd class only in well ventilated areas, and those of 3rd class only in surface blastings.

The blasting agents such as ANFO are more toxic than dynamite because they generate a larger proportion of nitrogen oxides. According to some investigations, the toxicity of NO_2 can reach up to 6.5 times more than that of CO for a given molar concentration.

In Spain, the limit concentration of gases for underground operations admissible for periods of 8 hours or less, are specified in the 'Complementary Technical Instructions: 04.7.02'.

REFERENCES

Blanc, J.P. & Thiard, R.: *L'Energie des Explosifs.* Explosifs, 1984.

Dick, R. et al.: *Explosives and Blasting Procedures Manual.* US Bureau of Mines, 1983.

Drury, F.C. & Westmaas, D.J.: *Considerations Affecting the Selection and Use of Modern Chemical Explosives.* SEE, 1980.

Du Pont: *Blaster's Handbook*, 16th Edition, 1980.

Elith, N.: *Measuring the Properties of Explosives.* Downline, ICI. September, 1986.

EXSA.: *Manual Prâctico de Voladura.* 1986.

Hagan, T.N.: *Explosives.* AMF. 1985.

Harries, G. & Beattie, T.: *The Underwater Testing of Explosives and Blasting.* Explosives in Mining Workshop. The Australian Institute of Mining and Metallurgy. 1988.

Lopez Jimeno, E.: Implantación de un Método de Cálculo y Diseño de Voladuras en Banco. Tesis Doctoral, ETS de Ingenieros de Minas de Madrid, 1986.

Mohanty, B.: *Energy, Strength and Performance and their Implications in Rating Commercial Explosives.* SEE, 1981.

Muñiz, E.: *Notas de Clase.* ETS de Ingenieros de Minas de Madrid, 1986. 'Tipos de Explosivos y Propiedades'. I Curso sobre Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez Pardo, Marzo 1986.

Union Española De Explosivos: *Explosivos y Accesorios.*

CHAPTER 11

Industrial explosives

11.1 INTRODUCTION

The industrial chemical explosives are classified in two large groups, according to their shock wave velocity.

a) *Rapid and detonating explosives.* With speeds between 2.000 and 7.000 m/s; and

b) *Slow and deflagrating explosives.* Under 2.000 m/s.

The deflagrating explosives include gunpowders, pyrotechnic compounds and propulsive compounds for artillery and fireworks, with practically no application in mining or civil engineering, with the exception of ornamental rocks.

The detonating explosives are divided into Primary and Secondary, depending upon their application. The Primary, for their high energy and sensitivity, are used as initiators for the Secondary, among which we can mention the compounds used in the blasting caps and cast primers (mercury fulminate, PETN, Pentolite, etc.). The Secondary are those that are applied to the breakage of rocks and, although less sensitive than the Primary, they do more useful work. These compounds can be mixtures of explosive substances or not, depending upon lower manufacturing price, on the better oxygen balance obtained and on the characteristics and properties that the ingredients give to the mixtures in the sense of sensitivity, density, strength, water resistance, etc.

The industrial explosives for civilian use are also divided into two large groups which, placed in order of importance of those most frequently used and not in that of their appearance on the market, are:

Blasting agents

These mixtures, with few exceptions, do not contain ingredients classified as explosive. The following are the most common:

- ANFO,
- ALANFO,
- Slurries or water gels,
- Emulsions,
- Heavy ANFO.

Conventional explosives

These are essentially made up of explosive substances that act as sensitizers of the mixtures. The best known are:

- Gelatin dynamites,
- Granular dynamites,
- Permissible explosives.

In this chapter the basic characteristics of each explosive, their composition and the influence of the different parameters upon their efficiency in rock blasting will be discussed.

11.2 DRY BLASTING AGENTS

This group includes, as indicated before, all those explosives that are not cap sensitive and do not include water in their composition. The common denominator in all dry blasting agents is Ammonium Nitrate, Fig. 11.1, some of whose properties are analyzed below.

11.2.1 *Ammonium Nitrate*

Ammonium Nitrate (NH_4NO_3) is a white inorganic salt with a melting point of 160.6°C. Alone, it is not an explosive, as it only takes on that property when it is mixed with a small quantity of combustible material with which it reacts violently by contributing oxygen. Compared with air which has 21% oxygen, NA has 60%.

Although NA can be found in diverse forms, in the manufacturing of explosives it is used in the form of small porous spherical prills, as it has the best characteristics for absorbing and retaining liquid fuels and is easily manipulated because of good pouring properties, without clogging or adhering.

The density of the loose-poured or bulk blasting-grade NA is approximately 0.80 g/cm^3, whereas the densities of the non-porous prills are close to those of the crystals (1.72 g/cm^3), but with lower values (1.40-1.45 g/cm^3) due to its microporosity. Higher density NA is not used as it absorbs less fuel substances, and therefore reacts more slowly in the detonating process.

Normally, the blasting-grade NA has a microporosity of 15% which, when added to the macroporosity reaches 54%.

The size of the particles usually vary between 1 and 3 mm.

When pure NA is heated to above 32.1°C, it changes crystalline shape:

β Orthorhombic	Crystal density = 1.72 g/cm^3
+ 32.1°C	
γ Orthorhombic	Crystal density = 1.66 g/cm^3

This transition is accompanied by a 3.6% increase in volume, with the consequent breakage of the crystals into

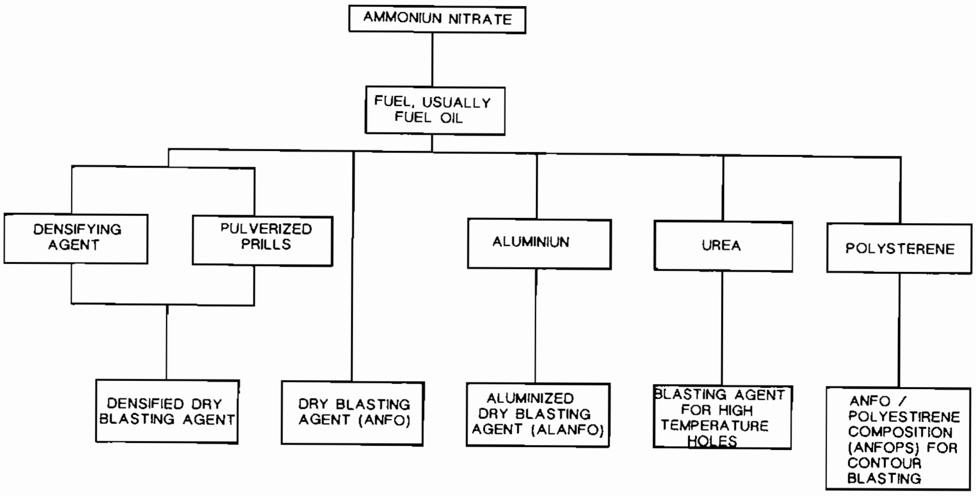

AMMONIUN NITRATE
FUEL, USUALLY FUEL OIL
DENSIFYING AGENT
PULVERIZED PRILLS
ALUMINIUN
UREA
POLYSTERENE
DENSIFIED DRY BLASTING AGENT
DRY BLASTING AGENT (ANFO)
ALUMINIZED DRY BLASTING AGENT (ALANFO)
BLASTING AGENT FOR HIGH TEMPERATURE HOLES
ANFO / POLYESTIRENE COMPOSITION (ANFOPS) FOR CONTOUR BLASTING

Fig. 11.1. Dry blasting agents with Ammonium Nitrate base.

Table 11.1.

Temperature of AN/air	Humidity at which rapid absorption occurs
10°C	76%
21°C	64%
32°C	59%

smaller ones. When the crystals cool and humidity is present, they tend to cake and form large lumps.

AN has great solubility in water which varies considerably at different temperatures:

At 10°C 60.0% solubility (weight basis),
At 20°C 65.4% solubility,
At 30°C 70.0% solubility,
At 40°C 73.9% solubility.

So from this one can see why ANFO is not used in wet blastholes.

AN is also highly hygroscopic, and may become liquid in the presence of air with a humidity above 60%. By adding inert hydrophilic substances such as powdered kaolin or clay, the absorbing of humidity by NA can be avoided, but its sensitivity is also lowered.

The temperature of the atmosphere also plays an important role in the absorption of humidity.

Sometimes, the grains of NA are coated with substances that avoid superficial moistening.

NA is completely stable at atmospheric temperature, but if it is heated above 200°C in a closed recepient, it may detonate. The presence of organic compounds accelerate the decomposition and lower the temperature at which this is produced. Therefore, with 0.1% of cotton, NA starts to decompose at 160°C.

11.2.2 *ANFO*

In 1947 a disastrous explosion of Ammonium Nitrate took place in Texas City (United States), as the substance was being protected with paraffin and only 1% of this constituted a good sensitizing fuel of AN.

Apart from the catastrophe itself, the attention of the manufacturers of explosives was drawn to the fact that NA had tremendous energetic potential, along with its low cost.

Any combustible substance can be used with AN to produce a blasting agent. In the United States at the end of the fifties carbon powder was used but, afterwards, it was substituted by liquid fuels as the mixtures were more homogeneous and intimate with AN. The most common product used is fuel oil which, when compared to other liquids such as gasoline, kerosene, etc., has the advantage of a higher point of volatility and, as a consequence, less risk of steam explosions.

Used oils were also tried as fuels, but they have the inconvenience of reducing the sensitivity to initiation and propagation, to detonation velocity and energetic yield. Owing to their high viscosity they tend to coat the surface of the AN prills, filling the macropores. For the moment, the use of sump oils is not justified either partially or totally from an economical point of view as they give more problems than advantages.

The fuel content plays a very important role in the different properties of ANFO. The decomposition reaction of the oxygen balanced system is:

$$3NH_4NO_3 + CH_2 \rightarrow 3N_2 + 7H_2O + CO_2$$

producing around 920 kcal/kg, that could be lower in commercial products depending upon the contents of inert materials, and a gas volume of 970 l. The stoichiometric mixture corresponds to 95.3% of AN and 5.7% of fuel oil, which is the equivalent of 3.7 liters of the latter for each 50 kg of AN.

The influence that the percentage of fuel has upon the energy released and the detonation velocity are indicated in Fig. 11.2.

It can be seen that neither higher nor lower percentages than indicated are of interest, if maximum yield is to be obtained in the blastings. On occasions, as in summer for example, it is usual practice to add more fuel oil to the ANFO, because up to 50% of the fuel can be lost because of the heat, with the subsequent reduction in efficiency. It is quite simple to control the quality of ANFO as it consists in extracting the fuel oil from a sample by using ether, Fig. 11.3, and taking the weight of the sample before and after the process.

The fuel contents also affect the quantity of harmful

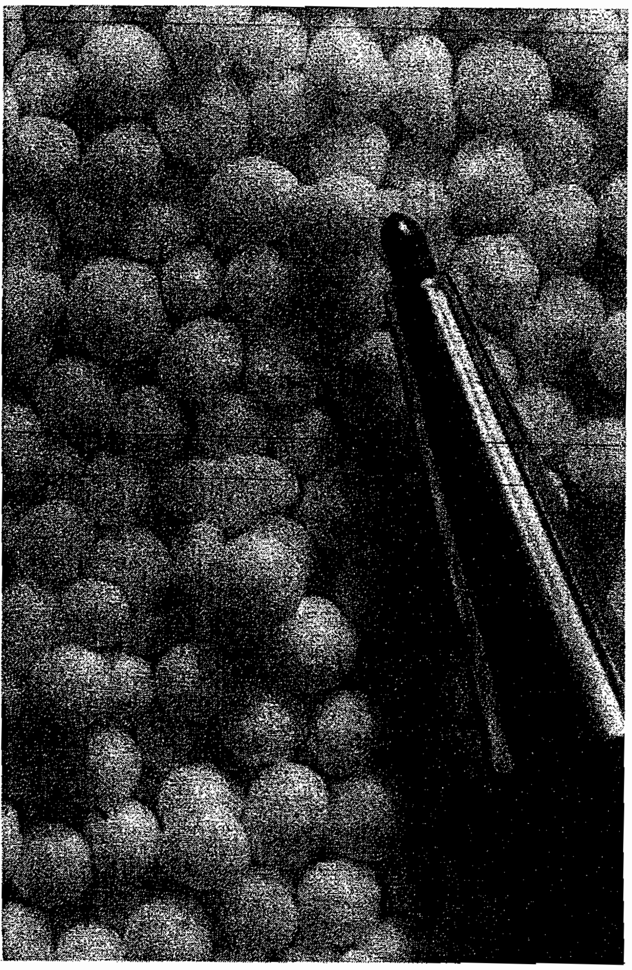


Photo 11.1. Porous Ammonium Nitrate prills.

gases released by the explosion (CO + NO), Fig. 11.4. When orange-colored fumes appear after a blast, this indicates an insufficient percentage of fuel oil, or that the ANFO has absorbed water from the blastholes or was not properly initiated.

The variation in sensitivity with the quantity of fuel is also noticeable because with 2% of fuel oil the initiation can be achieved with a blasting cap, although the available energy is quite low; and with an amount over 7% the initial sensitivity decreases substantially, Fig. 11.5.

As happens with AN, water is the main enemy of ANFO because it absorbs a large quantity of heat to produce steam and it considerably lowers the strength of the explosive. In 76 mm diameter charges, a humidity of more than 10% desensitizes the blasting agent, Fig. 11.6.

In these cases the only remedy is to wrap ANFO in waterproof sheaths or recipients.

The explosive characteristics of ANFO also vary with density. As this increases, the detonation velocity is also higher, but it is also more difficult to achieve initiation. Above a density of 1.2 g/cm^3 ANFO becomes inert and cannot be detonated or will only do so in the immediate area of the blasting cap.

The prill size of AN also has influence upon the density

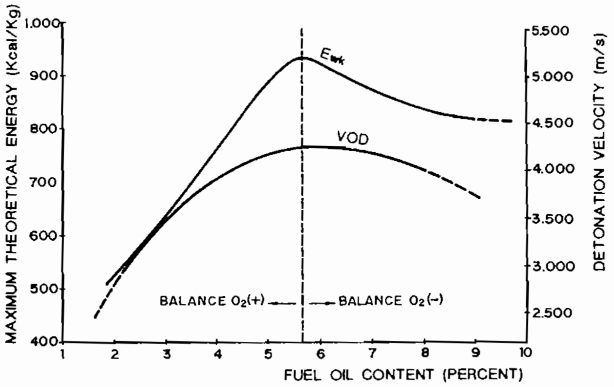


Fig. 11.2. Variation of thermodynamic Energy and detonation Velocity of ANFO with the fuel oil content.

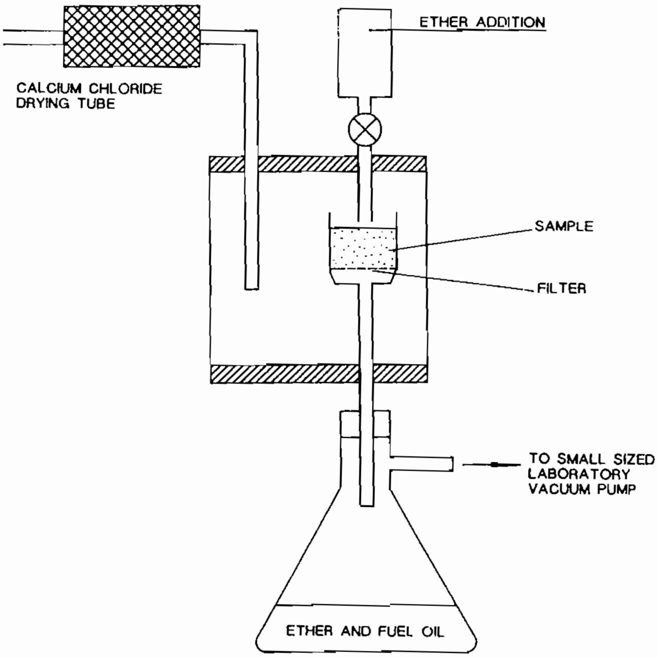

ETHER ADDITION
CALCIUM CHLORIDE DRYING TUBE
SAMPLE
FILTER
TO SMALL SIZED LABORATORY VACUUM PUMP
ETHER AND FUEL OIL

Fig. 11.3. Laboratory procedure to measure the percentage of fuel oil.

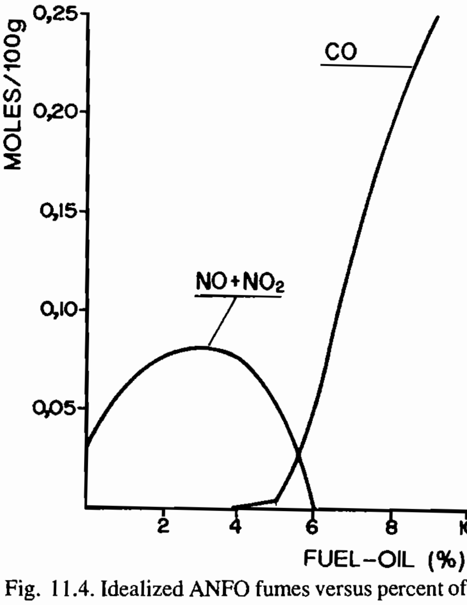


Fig. 11.4. Idealized ANFO fumes versus percent of fuel oil.

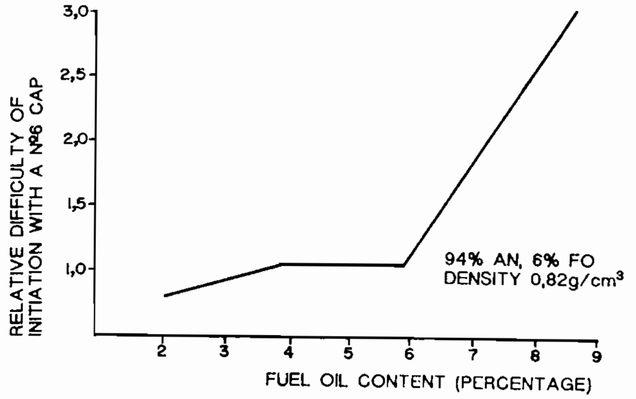


Fig. 11.5. Sensitivity to initiation of ANFO.

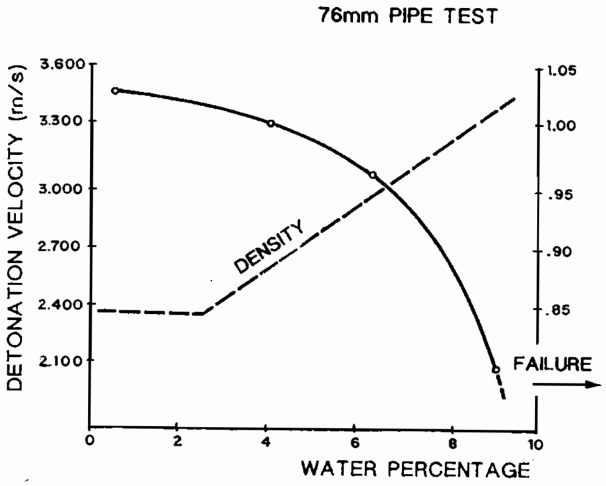


Fig. 11.6. Influence of water in ANFO performance.

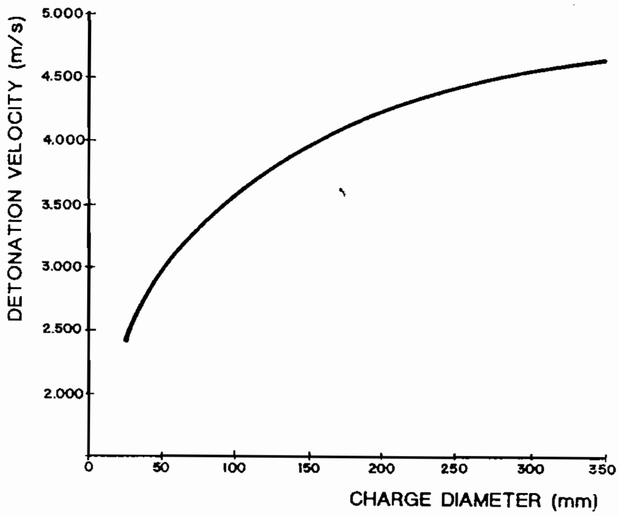


Fig. 11.7. Variation of VOD with charge diameter for confined full-coupled charges of bulk ANFO.

of the explosive. Therefore, when ANFO is reduced to under 100 mesh, its bulk density changes to 0.6 g/cm^3, which means that if a normal density is to be reached, between 0.8 and 0.85 g/cm^3, in order to obtain good detonation characteristics, it must be physically compressed by blow-loading.

On the other hand, the charge diameter is a design parameter that plays a decisive role in the detonation velocity of ANFO. Fig. 11.7.

The critical diameter of this explosive is influenced by its confinement and charge density. Used in blastholes with a loose-poured density of 0.8 g/cm^3, the critical diameter is around 25 mm, whereas with a density of 1.15 g/cm^3, it goes up to 75 mm.

The cap sensitivity of ANFO is reduced as the blasthole diameter increases. In practice, the primers of 150 g are effective in charge diameters under 150 mm; above this diameter, cast primers of 400 to 500 g are recommended.

Although ANFO is predominantely used as a bulk-loaded charge, it is important to know that the energy per meter of charge length decreases with decoupling. When the charge is not excessively confined, the VOD and peak pressure upon the blasthole walls are reduced.

11.2.3 *ALANFO*

As ANFO density is low, the resulting energy per meter of column length is also low. To elevate this energy, since 1968, products such as aluminum have been added with good technical and economical results, above all in strong, massive rocks and in high-cost drilling operations.

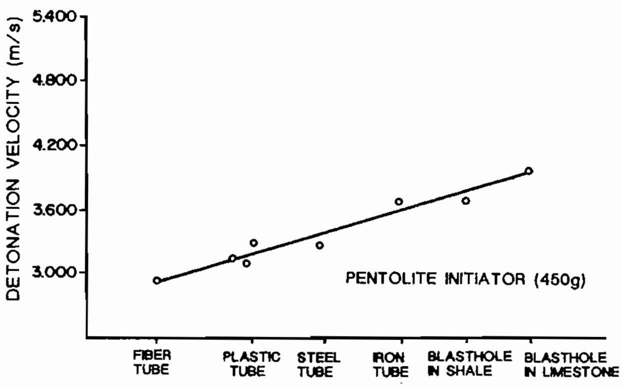


Fig. 11.8. Variation of VOD with confinement.

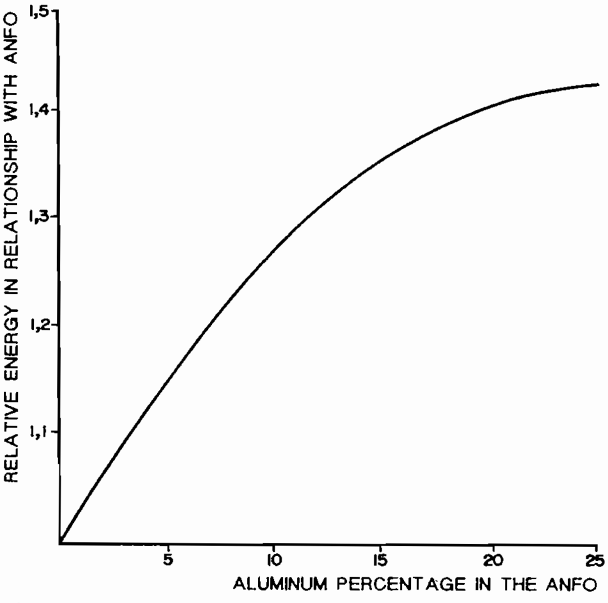


Fig. 11.9. Variation of energy yield of ALANFO with percentage aluminum.

When aluminum is mixed with ammonium nitrate in small quantities, the following reaction takes place:

$$2Al + 3NH_4NO_3 \rightarrow 3N_2 + 6H_2O + Al_2O + 1650 \text{ cal/g}$$

but if the percentage of aluminum is larger, the reaction produced is:

$$2Al + NH_4NO_3 \rightarrow N_2 + 2H_2 + Al_2O_3 + 2300 \text{ cal/g}$$

In Fig. 11.9, the energy produced by ALANFO with different quantities of aluminum is compared to that of ANFO.

The practical limit, as far as yield and economy go, is between 13 and 15 percent. Percentages above 25% cause a reduction in energy efficiency.

The aluminum should comply with the following specifications: almost 100% in size between 20 and 150 mesh, and be more than 94% pure.

In these blasting agents, the purity is not as critical as in slurries, because the galvanic reaction produced by the changes in pH are not a problem. The means that the leftovers or surplus aluminum from other processes can be used in the manufacturing of ALANFO.

11.3 SLURRIES

Slurries are blasting agents based on saturated aqueous solutions of AN, often with other oxidizers such as sodium nitrate and/or calcium nitrate, in which the fuels, sensitizers, gellants and cross-linking agents are dispersed to avoid the segregation of the solids.

These explosives were developed at the end of the fifties when Cook and Farnam obtained the first positive tests with a mixture of 65% of AN, 20% of AL and 15% water.

After these first results, Cook began to use TNT as a sensitizer and the patented commercial production began in Canada, spreading to the US.

Later on, the first tests were carried out with aluminium sensitized slurries. This metal brought on serious problems, as it reacted with water at atmosphere temperature giving off hydrogen. To avoid this phenomenon, the aluminium particles were protected with waterproof products.

In 1969, Dupont developed new slurries that did not contain traditional explosive compounds, nor particles of metals as fundamental sensitizers. They used organic substances as fuel such as those derived from amines, paraffins, sugars, etc.

In Fig. 11.10, the principal types of aqueous explosives obtained from Ammonium Nitrate, in two large groups, which are the slurries and the emulsions with their mixtures.

The manufacturing process of the slurries in use today is based upon the mixing of a solution of oxidizers with another of monomethyl amine nitrate (MMAN), and the addition of diverse solids and liquids such as oxidizers, thickeners, gellants, etc.

The solution of oxidizers is made up of water, ammonium nitrate and sodium nitrate. to which thiourea and part of the gums are added, giving a high viscosity in order to retain the gas bubbles. The sodium nitrate has the advantage of its large amount of oxygen and also the ability to lower the crystallization point of the saline solutions.

The MMAN solution is prepared by heating the containers in which it is transported, as it is in a solid state, because of its cristallization point between 33 and 39°C. This product has excellent sensitizing characteristics, as it is a very good fuel with a very negative oxygen balance and high density; apart from being quite insensitive to the dynamic subsonic shock and friction effects. The propor-

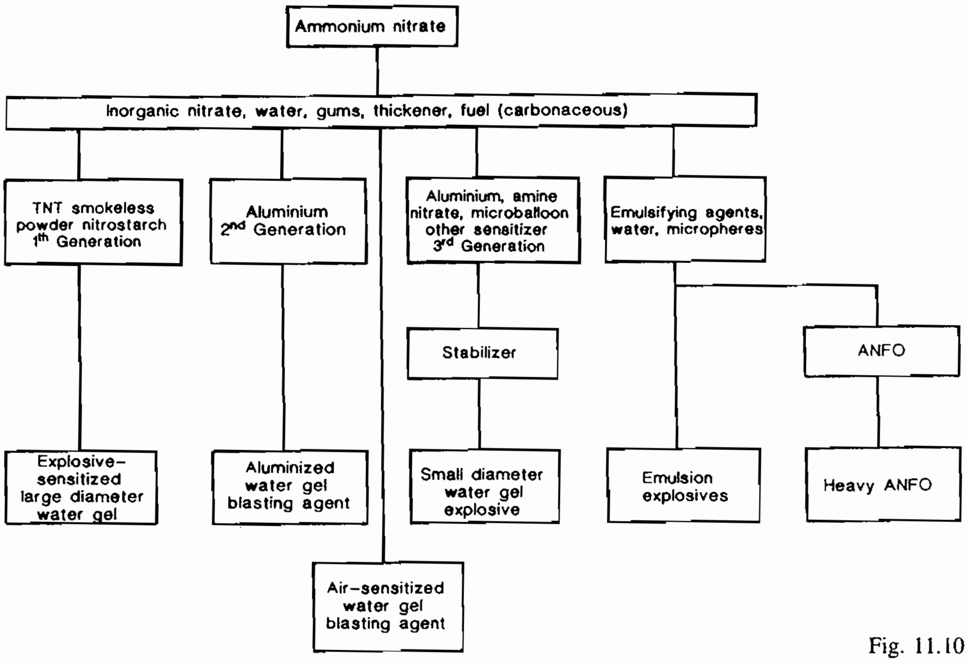

Ammonium nitrate
Inorganic nitrate, water, gums, thickener, fuel (carbonaceous)
TNT smokeless powder nitrostarch 1th Generation
Aluminium 2nd Generation
Aluminium, amine nitrate, microballoon other sensitizer 3rd Generation
Emulsifying agents, water, micropheres
Stabilizer
ANFO
Explosive-sensitized large diameter water gel
Aluminized water gel blasting agent
Small diameter water gel explosive
Emulsion explosives
Heavy ANFO
Air-sensitized water gel blasting agent

Fig. 11.10. Aqueous blasting agents made from AN.

tions of MMAN in the slurries varies between 10 and 35%.

The mixture of the solid aditives is made up of aluminium, nitrostarch, gums and other substances in smaller proportions.

The aluminium proportionately increases the sensitivity of the slurries and the gums, and the nitrostarch serves to thicken the mixtures. Occasionally cross-linking agents are added to gell the slurries.

On the other hand, as the percentage of water used is unsufficient to disolve all the nitrates, a certain amount of these are added in a solid state to form part of the disperse phase.

In order to change the density, chemical gasification (bubbles) can be carried out, usually with sodium nitrite, or the addition of low density products, glass microballoons, etc.

The mixture of all these components is carried in a continuous or discontinuous fashion, using mixing tanks that can be agitated and can be installed in static plants or upon trucks.

One of the characteristics of slurries is that they are very safe, to manufacture as well as to handle, because the sensitizers in their composition are not essentially explosives. Even so, their aptitude for detonation is very good which means that slurries can be used in small diameters and be initiated with conventional blasting caps.

Their water resistance is excellent and their strength, a fundamental application characteristic, is equivalent or above that of conventional explosives and can be adjusted as a function of the composition of the slurry. The energies developed oscillate in the range of 700 to 1.500 cal/g.

The density can also be modified, from 0.8 to 1.6 g/cm^3, starting from a basic value of between 1.4 and 1.5. By adding chemical gasifiers, as mentioned before, or low density additives, this parameter can be reduced. These reductions have influence on the explosives, increasing the detonation velocity in most instances, as well as the sensitivity, Fig. 11.12.

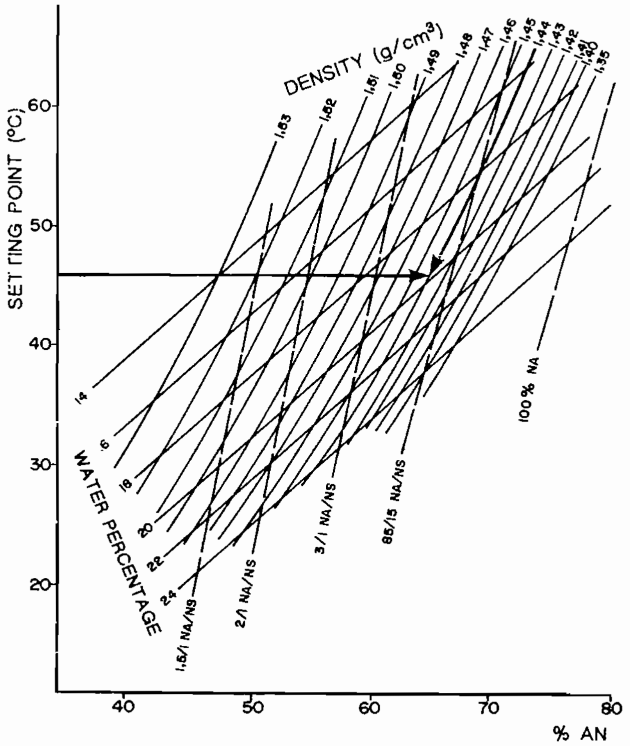


Fig. 11.11. Setting point of the AN/SN/water system, depending upon its composition and density.

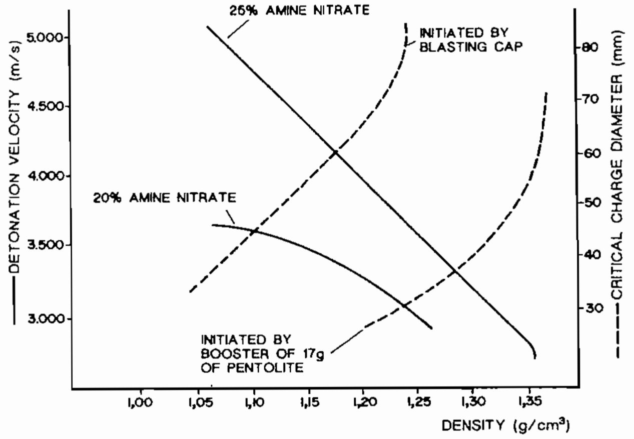


Fig. 11.12. Watergel sensitivity and detonation velocity as a function of changing density.

Obviously, the variety of products that can be obtained with different compositions is very large. From the cartridged slurries, similar to conventional gelatin explosives, up to those that can be pumped which have rheological characteristics that allow them to be treated as fluids. In the latter case, the advantages are that charging is mechanized and the blastholes are totally filled.

Slurries that are sensitized with aluminium have better quality fumes in blasting than those from conventional explosives.

11.4 EMULSIONS

This group of explosives, the newest on the market, maintain the properties of the slurries but at the same time improve two fundamental characteristics, strength and water resistance.

The interest in these products started in the sixties, when the basic requirement of an explosive was being investigated: the combination of an oxidizer with a mineral oil in a detonation reaction. These products have remained chemically invariable over the years (ammonium nitrate + fuel oil), but their physical form has changed drastically.

Table 11.2 summarizes in chronological order of aparition of the explosives, the oxidizers, fuels and sensitizers used in the manufacturing of each one.

From the chemical point of view, an emulsion is a two-phase system in which an inner or dispersed phase is distributed in an outer or continuous phase.

The explosive emulsions are of a so called *water in oil* type, in which the aqueous phase is composed of inorganic oxidizing salts dissolved in water, and the oil phase of

Table 11.2.

Explosive	Oxidizer	Fuel	Sensitizer
Dynamite	Solid Nitrate salts	Solid Meal absorbents	Liquid Nitroglycerine, voids/bubbles friction
ANFO	Solid Nitrate salts	Liquid Diesel oil	Voids/friction
Slurry	Solid/liquid Salt solutions, nitrate salt	Solid/liquid Aluminium, carbonaceous	Solid/liquid TNT EGMN Fine aluminium/ bubbles
Emulsion	Liquid Salt solution	Liquid Oils, waxes	Bubbles

Table 11.3. Characteristic sizes of oxidizers in explosives (Bampfield & Morrey, 1984).

Explosive	Size (mm)	Form	VOD (km/sec)
ANFO	2.000 mm	All solid	3.2
Dynamite	0.2 mm	All solid	4.0
Slurry	0.2 mm	Solid/liquid	3.3
Emulsion	0.001 mm	Liquid	5.0-6.0

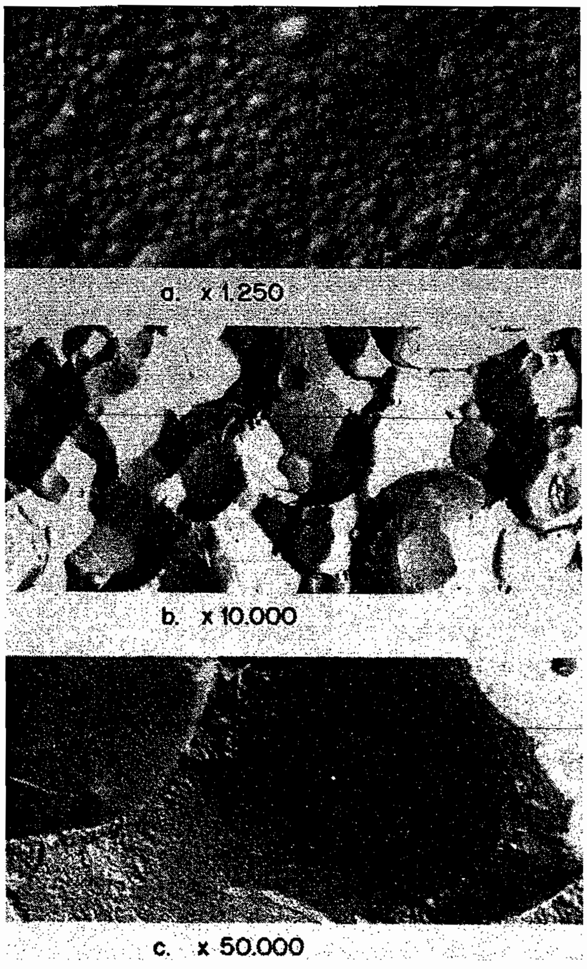


Photo 11.2. Structure of the emulsions (Courtesy of Bampfield and Morrey, 1984)

a liquid fuel immiscible with the water that is hydrocarbonated.

During the development of the explosives, there has been a progressive reduction in particle size, going from solids to saline solutions with solids and, lastly, to the droplets of an explosive emulsion. Table 11.3.

Thus, it can be understood that the difficulty in manufacturing emulsions can be found in the oil phase because in the final oxygen balance a 6% of the emulsion weight, which is the oil, must surround the other 94% that is in droplet form.

In the previous Table, the detonation velocities of each of the explosives, which correspond to a given diameter, reflect the strong dependency of the efficiency of the reaction upon the particle size.

The structure of the emulsions can be seen in the three photographs that follow, where the droplets of saturated solution (oxidizing) assume a polyhedral shape, not spherical, and are coated with a film of continous fuel phase. In Photo 11.2c, the size of the droplets in comparison with that of a prill of ammonium nitrate is 100 times smaller.

In order to obtain an adequate sensitization of the explosives, when these do not contain liquid or solid chemical sensitizers, a physical mechanism such as gas bubbles that when adiabatically compressed produce the *hot spot* phenomenon, favoring initiation as well as detonation propagation.

The gassifying agents are made up of expanded polystyrene beads or glass microballoons.

As to types of emulsion, there are many products under that name that have different properties related to the characteristics of the continuous phase and their effect on viscosity and consistency.

Depending upon the type of fuel, fuel-oil, waxes, gums, etc., the rheological characteristics of the emulsions are different, as well as their applications and methods of handling. Also the type of emulsifying agent used to reduce the superficial tension between the two immiscible liquids and therefore permit the formation of

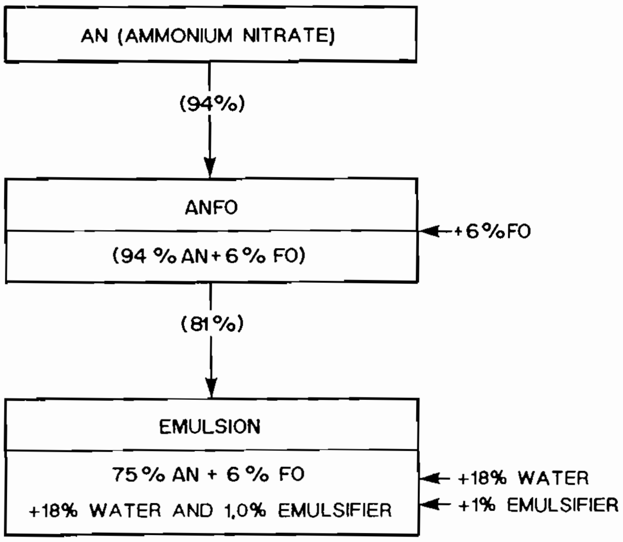


Fig. 11.13. Basic composition of an emulsion.

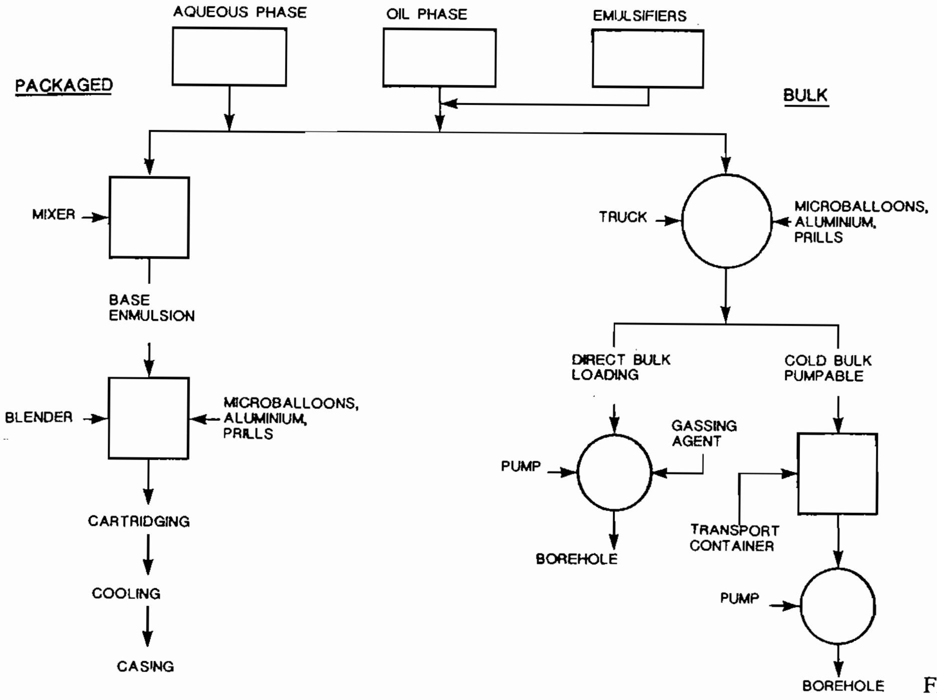


Fig. 11.14. Production pattern for emulsions.

the emulsion, can help to avoid problems of coagulation in large drops of the ammonium nitrate solution as well as the phenomenon of crystallization of the salts.

Another aspect that has to be taken into account is the cooling of the product from the moment of its manufacturing, which is carried out at a temperature close to 80°C, until the instant of its use.

The preparation pattern of the emulsions, either cartridged or bulk, is given in Fig. 11.14. Starting from the different components: aqueous oxidizing phase, oil and emulsifying-stabilizing phase, and the previous heating of these, an intense dynamic agitation follows to obtain a base emulsion that is later refined in order to homogeneize and stabilize it. Afterwards, it is mixed with the dry products that are added to adjust the density or the strength of the explosive. These solid products can be powdered aluminium, gasifying agents to reduce density, ammonium nitrate prills, etc.

Aluminium powder, although increasing the energy developed by the explosive, has a reducing effect upon the detonation velocity.

On the other hand, the sensitivity of the emulsion decreases as the density increases, making it necessary to work above the critical diameter and utilize powerful initiators.

The present tendency towards using emulsions in rock breakage is based upon the numerous advantages given below:

– Lower cost, as its manufacturing does not require the use of high-priced gums and starches.

– Excellent water resistance.

– Possibility of obtaining products with densities of between 1 and 1.45 g/cm^3.

– High detonation velocitied, 4000 to 5000 m/s, with little effect of the cartridge diameter.

– Very safe to manufacture and handle.

– Possibility of mechanized charging and preparation of mixtures with ANFO.

On the other hand, its problems are due to the strict preparation conditions, its alterability under low temperatures, its contamination during charging if it is used in bulk form, storing time and prolonged transportation periods.

11.5 HEAVY ANFO

In the present technology of blastings it is unquestionable that ANFO constitutes the basic explosive. Diverse attempts have been made to obtain more energy from this explosive, going from the crushing of high-density ammonium nitrate prills to the use of high-energy liquid fuels such as nitroparaffins, methanol and nitropropane, but they have not prospered commercially.

Heavy ANFO, which is a mixture of base emulsion with ANFO, gives a new perspective to the field of explosives.

ANFO leaves interstitial voids which can be occuied by a liquid explosive such as an emulsion that acts like an energizing matrix, Fig. 11.15.

Although the properties of this explosive depend upon the percentages of the mixture, its main advantages are:

– More energy,

– Better sensitivity,

– High water resistance,

– Possibility of charging with variation in energy along the length of the blasthole.

The manufacturing is relatively simple, as the emulsion matrix can be prepared in a static plant and transported by truck to a storage tank, or be pumped into a mix truck. With these trucks, the mixtures of emulsion with ammonium nitrate and fuel oil in adequate proportions

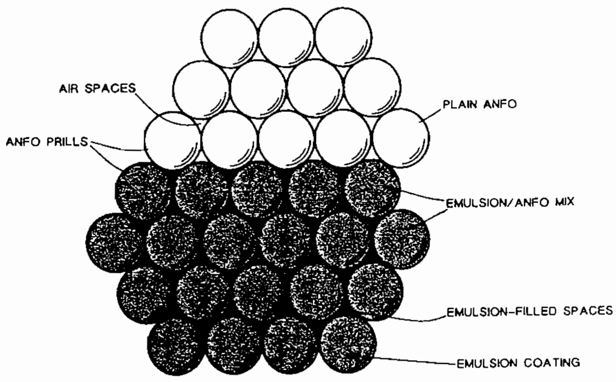


Fig. 11.15. Mix of emulsion and ANFO

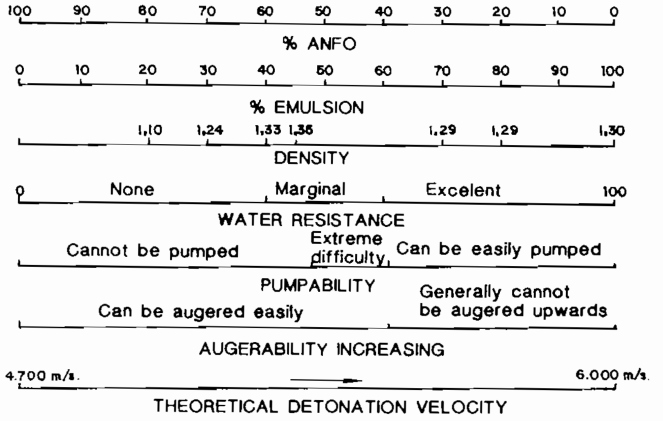


Fig. 11.16. Charging and water resistance characteristics of different types of heavy ANFO (Du Pont, 1986).

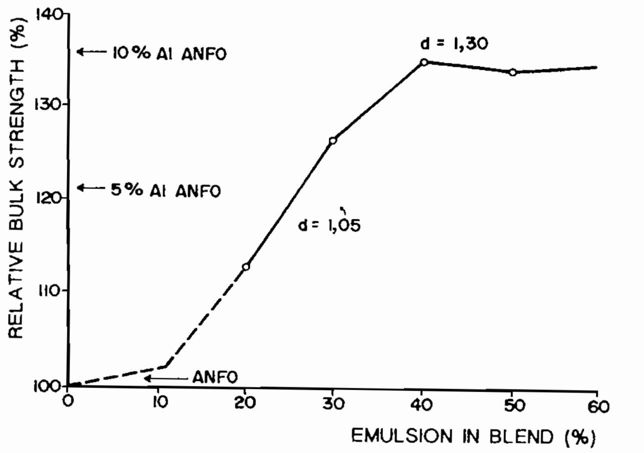


Fig. 11.17. Effect of matrix level on strength (Bampfield and Morrey, 1984).

for the work at hand can be prepared in-situ. Fig. 11.16.

In Fig. 11.17, a variation of the Relative Volume Strength (ANFO = 100) is shown for Heavy ANFO in function with the percentage of emulsion.

It can be seen that a Heavy ANFO 70/30 has more strength than an ALANFO of 5%, and a mixture of 60/40 is almost comparable to an ALANFO of 10%. Curiously, when the emulsion matrix increases beyond 40%, the strength diminishes due to excessive separation of the particles of ANFO which hinders their performance as hot spots and propagators of the shock wave.

The density of the mixture increases with the percentage of emulsion, reaching the maximum energy with an emulsion value of 1.3 g/cm^3 approximately.

In Fig. 11.18, the variation in sensitivity of Heavy ANFO with the increase of emulsion percentage is indicated. The sensitivity decreases with an increment in density, requiring a stronger blasting cap each time. For a density of 1.33, a 450 g cast primer of Pentolite, minimum, is necessary.

With the relatively recent acceptance of heavy ANFO in the industry, an aluminized heavy ANFO with very high bulk strengths for relatively low cost also has potential for improved mining efficiency and cost savings for certain operations.

Aluminum increases total energy output, bulk strength and detonation temperature and pressure of the explosive but does not greatly affect the velocity of detonation. The relationship for aluminum addition to a 30/70 (emulsion/ANFO) heavy ANFO is shown in Fig. 11.19.

Table 11.4 lists the bulk strengths of ANFO, emulsion and several heavy ANFOs produced using low density porous prilled ammonium nitrate, with varying percentages of aluminum addition.

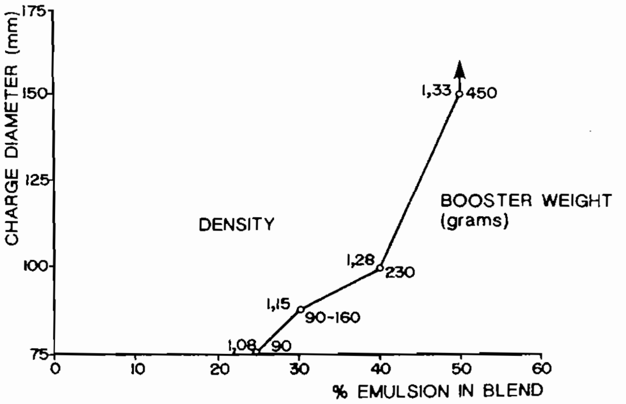


Fig. 11.18. Effect of matrix level on density, critical diameter and minimum booster (Bampfield and Morrey, 1984).

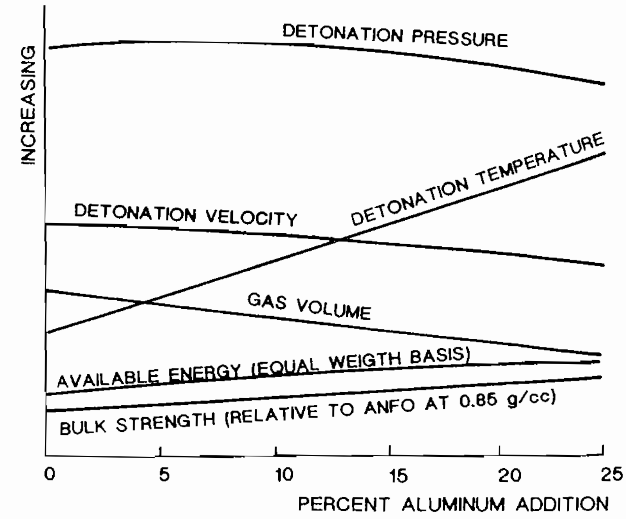


Fig. 11.19. Effect of increasing aluminum addition to a 30% emulsion Heavy ANFO on different characteristic properties.

Table 11.4.

Explosive	1	2	3
ANFO	0,85	100	1.00
Al/ANFO (5% Al)	0.88	112	1.16
Al/ANFO (10% Al)	0.91	123	1.32
Al/ANFO (15% Al)	0.94	134	1.48
NCN EMULSION (0% Al)	1.15	78	1.06
NCN EMULSION (5% Al)	1.21	91	1.30
NCN EMULSION (19% Al)	1.27	103	1.54
NCN EMULSION (15% Al)	1.30	117	1.79
ANFO + 10% EMULSION (0% Al)	0.93	98	1.07
ANFO + 20% EMULSION (0% Al)	1.01	96	1.14
ANFO + 30% EMULSION (0% Al)	1.11	93	1.21
ANFO + 40% EMULSION (0% Al)	1.20	91	1.28
ANFO + 50% EMULSION (0% Al)	1.29	89	1.35
ANFO + 30% EMULSION (5% Al)	1.14	105	1.41
ANFO + 30% EMULSION (10% Al)	1.16	116	1.58
ANFO + 30% EMULSION (15% Al)	1.19	127	1.78

1. Density (g/cm^3). 2. Relative weight strength ANFO = 100. 3. Bulk strength relative to ANFO at 0.85 g/cm^3 density (ANFO = 1.00). Source: Crosby & Pinco (1990).

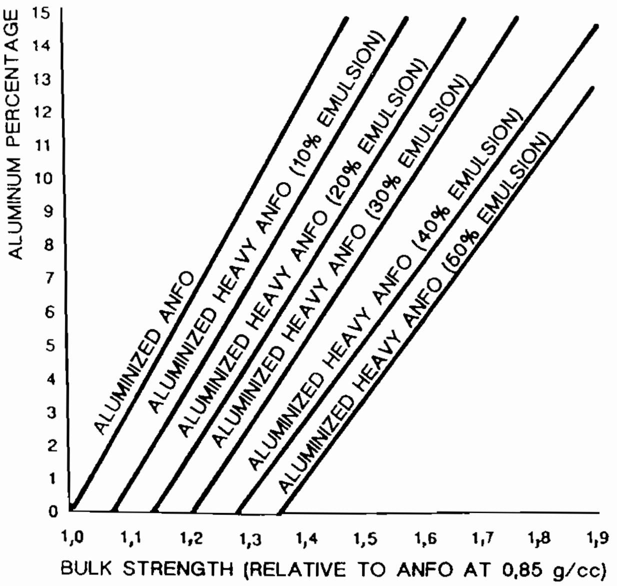


Fig. 11.20. Percent aluminum content at various bulk strengths for selected AL/ANFO and Heavy ANFO formulations.

The reaction of aluminum during detonation results in the formation of solid oxides, and no gaseous products are formed containing aluminum. The volume of gas generated from the total explosive is therefore reduced. The heat of formation of the aluminum oxide is very high, 16.260 kj/kg, and results in a considerable gain in the heat of the explosion which increases the temperature of the gases. This higher gas temperature helps to offset the reduction in gas volume as a given volume of gas can do more work as the temperature is increased. The overall outcome is that aluminum addition allows more work to be done by the same volume of explosive, thus permitting drillhole spacings and burdens to be increased while producing corresponding improvements in blast fragmentation.

Fig. 11.20 allows the actual optimum explosive formulation to be determined. It presents the aluminum addition on a weight percent basis for each of the dewatered hole explosive products with bulk strengths between 1.0 and 1.9.

11.6 GELATIN DYNAMITES

In 1875, Alfred Nobel discovered that a large quantity of nitroglycerine (NG) could be disolved and be retained in nitrocellulose (NC), obtaining a product with plastic consistency of easy use and handling for those times. That gelatin dynamite made up of 92% NG and 8% NC had a zero oxygen balance and released even more energy than pure NG.

Afterwards, with the intention of reducing the strength of that explosive mixture, oxidizing and fuel substances were added, in proper proportion to maintain the oxygen balance. Apart from reducing the costs of manufacturing, the gelignite consistancy was conserved. Therefore, the NC-NG percentage of the present day gelatin dynamites oscillates between 30 and 35% and the rest corresponds to oxidizers such as ammonium nitrate, to the fuels and to other special products that correct the hygroscopicity of the nitrates. Even though the percentage of NG is small, the resulting strengths are not as low as would be expected, reaching levels close to 80% of straight gelatin dynamite.

The principal advantages of these explosives, which have been widely used up until recently, are:

- High strengths.
- High densities, from 1.2 to 1.5 g/cm^3.
- High detonation velocities, between 5000 and 6000 m/s.
- Great water resistance and chemical stability.

Its most important problems are:

- Risk of accidents in manufacture and transportaion.
- Sensitivity to subsonic stimuli and, as a conseuence, quite dangerous if the machinery hits or comes into contact with left over explosive.
- Gives headaches because NG dialates the blood vessels.
- Lack of flexibility in use under extreme atmospheric conditions.
- High manufacturing costs.

11.7 GRANULAR DYNAMITE

Those explosive mixtures that are sensitized with NG but with a percentage under 15% have a granular or powdery consistency.

In this group of explosives, those having an inert base and those with an active base should be distinguished. The first, now obsolete, were developed by Nobel in 1867 and consisted of NG and kieselguhr or calcinated infusorian earth. Those with an active base are manufactured by substituting the inert materials for a mixture of oxidizers and fuels which give added strength.

The first oxidizer used was sodium nitrate, which was later replaced by ammonium nitrate, of more efficient energy. In this case special additives were also used to reduce the hygroscopicity of AN.

In other granular explosives, part of the NG is totally or partially replaced by TNT.

The characteristics of these explosive mixtures are:

- Lower strength than the gelignites.
- Detonation velocities and densities under 3000 to 4500 m/s, and 0.9 to 1.2 g/cm^3, respectively.
- Very little water resistance.
- Adequate for soft and semi-hard rock as a column charge.

11.8 PERMISSIBLE EXPLOSIVES

Permissible explosives are those designed for use in underground coal mines where the presence of explosive gases or dust is dangerous for normal blasting. Their main characteristic is the low explosion temperature.

At present, the Permissible Explosives are classified in two groups. The first is that in which the composition has an additive that takes the role of explosion inhibitor, usually sodium chloride that, depending upon its distribution, percentage, etc. increases, in more or less degree, the safety in an inflammable atmosphere.

The second and more recent group is called Reinforced Safety or of ionic exchange and obtains lower explosion temperature with diverse ingredients that react at the moment of detonation, forming an inhibitor at that same instant. These explosives are usually made up of a small percentage of NG, a fuel and the par saline sodium nitrate-ammonium chloride. The reaction that takes place is:

$$NaNO_3 + NH_4Cl \rightarrow NaCl + NH_4NO_3$$

The ammonium nitrate acts afterwards as an oxidizer and the sodium chloride in an incipient state has great cooling power, much more than in the conventional permissible explosives.

If by chance, a cartridge of ionic exchange detonates in the open or under weak confinement conditions, the phenomena that take place are the explosive decomposition of the nitroglycerine and the inhibiting action of the ammonium chloride as the reaction of the saline par is not produced. In any case, deflagration is avoided which would be very dangerous in an inflammable atmosphere.

The practical characteristics of permissible explosives are a medium or low strength, detonation velocities between 2000 and 4500 m/s, densities between 1 and 1.5 g/cm^3 and poor water resistance, with few exceptions.

11.9 BLACKPOWDERS

Present day blackpowder for mining uses has the following composition: potassium nitrate (75%), sulphur (10%) and carbon (15%). It is always in granular and grafited form with sizes that oscillate between 0.1 mm and 4 mm, and generally comes in bags of 1, 2.5 and 5 kg.

The combustion velocity depends upon the density of the blackpowder and the confinement conditions, and is always under 2000 m/s, which obviously means it is a deflagrating explosive.

It strength with respect to straight gelatin dynamite is around 28%, and the specific energy of 23.800 kgm/kg, with a maximum temperature of about 200°C. It has very poor water resistance.

Nowadays, the use of blackpowder is limited to the extraction of ornamental blocks and the breakage of

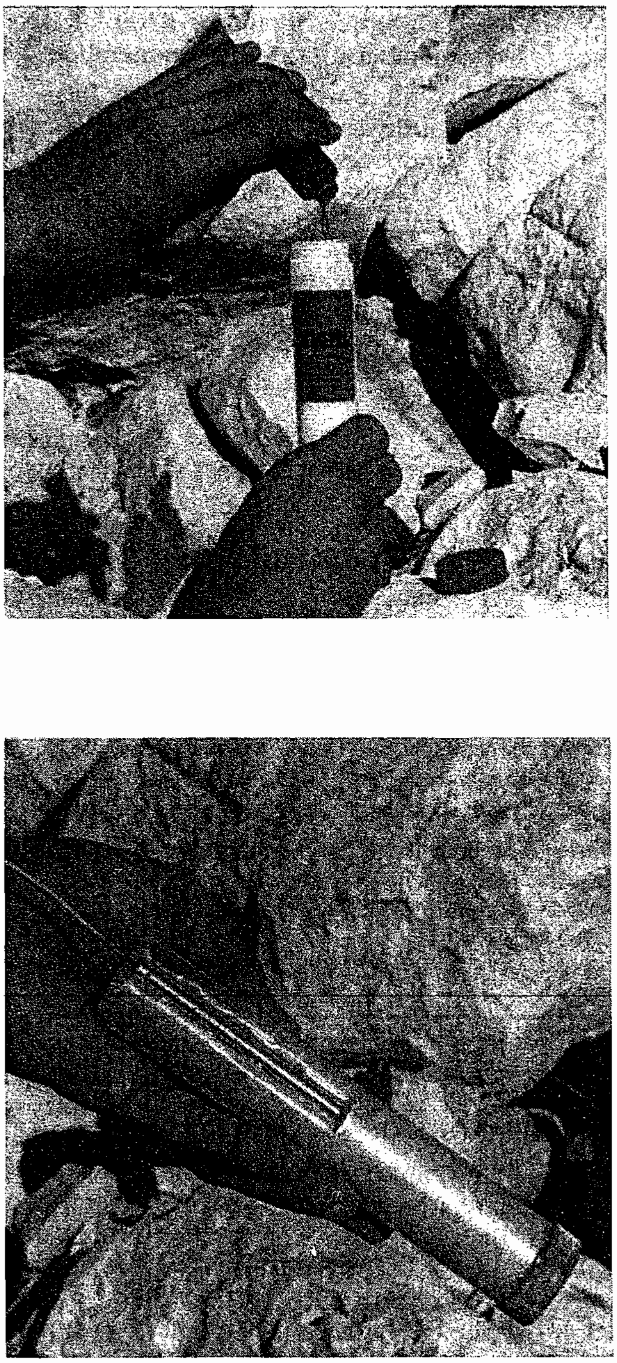


Photo 11.3. Preparation of a binary explosive cartridge (Courtesy of Kinepak, Inc.).

Table 11.5.

	Commercial denomination	Relative strength (%)	Density (g/cm^3)	Velocity of detonation (m/s)	Specific energy (kgm/kg)	Water resistance	Applications
ANFOS	Nagolita	65	0.80	2.000	94.400	Bad	Opencast operations, in the absence of water. Column charging.
	Alnafo	75	0.80	3.000	96.100	Bad	Blasting of soft and medium rocks.
	Naurita	65	0.80	2.000	94.320	Bad	Designed for blastholes with high temperatures.
Water gels	Riogel 2	85	1.17	4.500	94.801	Excellent	Blasting in presence of water. Base charge in open cast and quarrying. Tunelling and shaft sinking.
	Riogur	73	1.1	3.300	88.813	Excellent	Profilling and presplitting blasts.
Emulsions	RIOMEX E20-24 (Cartridged) Series E20-E24	103-145*	1.15	4.500-5.200	–	Excellent	Underground and surface blasting. Excavation of ditches, trenches and general civil works. Wet blastholes.
	RIONEX V2 (Bulk) Series V20-V24	109-154*	1.23	5.000-5.400	–	Excellent	Open pit mining. Large quarries. Public works. Able to use with automatic loading sistems.

*RBS = Relative Bulk strength % (RBS ANFO = 100).

Table 11.6.

	Commercial denomination	Relative strength (%)	Density (g/cm^3)	Velocity of detonation (m/s)	Specific energy (kgm/kg)	Water resistance	Aplications
Gelatinous	GOMA 1ED	90	1.49	6.000	104.158	Very good	Blasting of very hard rocks.
	GOMA 1-AGV	80	1.55	7.000	96.800	Very good	Blasting under high pressures. Seismic prospections.
	GOMA 2EC	85	1.40	5.250	102.400	Good	Blasting unnder presence of water. Base charge: combined with blasting agents. Hard rocks. Underground and surface blasts.
Powder explosives	AMONITA 2I	77	0.95	3.000	87.500	Poor	Blasting of soft to medium hard rocks.
	LIGAMITA 1	77	1.10	3.300	86.400	Poor	Blasting of soft to medium hard rocks. Used in underground mining and quarrying as well as in underground operations.
Permitted explosives	EXPLOSIVO DE SEGURIDAD 9	45	1.50	4.500	49.150	Good	Used in hard rocks and coal blasting operations.
	EXPLOSIVO DE SEGURIDAD 20 SR	40	1.15	2.500	36.650	Poor	Used in underground coal blasting operations.
	EXPLOSIVO DE SEGURIDAD 30 SR	35	1.10	2.000	30.190	Poor	Used in underground coal blasting operations and soft rocks.

elastoplastic materials such as gypsum, which break better under the continuous effect of the gases than by instantaneous strain. Therefore, the idea is to take advantage of the great thrust of the gases, instead of the breaking effect which is low.

11.10 TWO-COMPONENT EXPLOSIVES

Individually, the components of two-component explosives, also called binary explosives, are not classified as explosives. When shipped and stored separately they are not normally regulated as explosives, but they should be protected from theft.

The most common two-component explosive is a mixture of pulverized ammonium nitrate and nitromethane, although other fuel sensitizers such as rocket fuel have been used. The components are carried in separate containers to the jobsite, where the container of liquid fuel is poured into the ammonium nitrate container. After the prescribed waiting time the mixture becomes cap sensitive and is ready for use.

Two-component explosives are sometimes used where only small amounts of explosives are required such as in powerline installation and light construction. Where large amounts of explosives are needed, the higher cost per kilo and the inconvenience of onsite mixing negate the savings and convenience realized through less stringent storage and distribution requirements.

11.11 EXPLOSIVES COMMERCIALIZED IN SPAIN

In Tables 11.5 and 11.6, the technical characteristics of the commercialized explosives in Spain by UEE are given.

REFERENCES

Anonymous: *The Story is Dynamite.* Australian Mining. July, 1981.

Bampfield, H.A. & Morrey, W.B.: *Emulsion Explosives* CIL, Inc. September, 1984.

Blanco, R.: *Explosivos de Seguridad – Nuevas Tendencias.* Jornada

Técnica. Medidas de Seguridad en el Uso y Manejo de Explosivos en Excavaciones. INSH. October, 1986

Brulia, J.C.: *Power AN Emulsion/ANFO Explosives System*. SEE. 1985.

Buchta, L.: *Slurry Explosives* Mining Magazine, May, 1972.

Cook, M.A.: *How Dry Mix Explosives Can Increase Costs – Even in Dry Holes*. E/MJ, September, 1971.

Chaver, W. et al.: *Efectos de las Variaciones del Balance de Oxígeno y de la Granulometría en las Propiedades de los Explosivos Secos*. Jornadas de Especialistas en Tronaduras, Chile, 1983.

Chironis, N.P.: *Search for Better Blasting*. Coal Age. July, 1982.

Chironis, N.P.: *Emulsified Blasting Agents Boost*. Coal Age. January, 1985.

Dannemberg, J.: *Open Pit Explosives*. E/MJ. July, 1982.

Dick, R. et al.: *Explosives and Blasting Procedures Manual*. US Bureau of Mines. 1983.

Favreau, R.F. et al.: *Exhaust Temperature – of Explosion as a Criterion for Predicting Fire Hazards Due to Different Blasting Explosives*. CIMM. May, 1986.

Hagan, T.N.: *Explosives and Initiating Devices*. Australian Mining. July, 1981.

Herries, G.: *Explosives*. AMF, 1977.

Heltzen, A.M. & Kure, K.: *Blasting with ANFO/Polystryene Mixtures*. SEE, 1982.

Le Couteur, L.: *Emulsion Slurry Explosives – The New Generation of Explosive Products*. 4th Open Pit Operator Conference. 1982.

Mairs, D.B. & Tupling, R.V.: *Application of Emulsion Explosives in Underground Mining*. CIL, Inc. 1985.

Manon, J.J.: *Explosives: Their Classification and Characteristics*. E/MJ. February, 1977.

Muñiz, E.: *Apuntes de Explosivos*. ETS. Ing. de Minas, 1986.

Rios, J.: *Aplicicación de los Slurries al Arranque de Rocas*. C y E. December, 1981.

Tuttle, C.E.: *Small Diameter Slurries*. Pit & Quarry, 1972.

Thornley, G.M. & Fuk, A.G.: *Aluminized Blasting Agents*. IRECO Chemicals.

Union Española de Explosivos: *Empleo del Riogel en Voladuras de Interior*. Jornadas Técincas, UEE. *Explosivos y Accesorios. Emulsiones – Explosivas*. Jornadas Técnicas, UEE. *Uso de los Riogeles Vertibles en Explotaciones a Cielo Abierto*. Jornadas Técnicas, UEE.

CHAPTER 12

Explosive selection criteria

12.1 INTRODUCTION

One of the groups of parameters that can be controlled by the technicians in blastings is that of the explosives. The selection of the type of explosive forms an important part of the blast design and, consequently, the obtainable results.

Those who use explosives quite often fall into a routine and into the mirage of minimum costs of breakage without taking into account a series of factors that must be analyzed for proper selection: cost of the explosive, charge diameter, rock characteristcs, volume of rock to be blasted, presence of water, safety conditions, explosive atmospheres, and supply problems.

12.2 EXPLOSIVE COST

The cost of the explosive is obviously a very important selection criterium. To start with, one must choose the lowest cost explosive with which the work at hand can be carried out.

The comparative costs of the explosives per unit of weight, with Ammonium Nitrate as a reference, are shown in Fig. 12.1, drawn up from that of Wright (1986).

It can be seen that the least costly explosive is ANFO, and has a frequency of use rating of 50 to 80%, varying in different countries. Other attractive features of this blasting agent is its safeness, ease of storage, transportation and handling, as well as the possibility of bulk charging.

However, even though the cost is low, ANFO has some problems, such as poor water resistance and low density.

When speaking of cost, it would be more correct to do this by expressing it per unit of available energy ($/kcal) than by unit of weight ($/kg) because, the fact is that the results of the blastings depend upon the energy destined to fragmentation and swelling of the rock.

On the other hand, it must not be forgotten that the objective of blasting is to carry out breakage at the lowest possible cost and that in hard rocks, drilling is a burdensome task that could well compensate the use of costly explosives that have more strength, or selective charges made up of a high density – strength bottom charge and a lower density, medium strength column charge, Fig. 12.2.

For a blast design using a fixed hole size requiring an explosive or explosives of a particular bulk strength, and having fixed collar and subgrade dimensions, the lowest blast cost will be achieved by using the explosive having the required bulk strength at the lowest cost per unit length of loaded blasthole. The possible choice of product is shown in Fig. 12.3, which presents the relationship between relative weight strength and relative bulk strength for different aluminized and non-aluminized ANFO, emulsion and heavy ANFO systems. This figure also indicates which products can be used in wet, dry or dewatered blastholes. For a particular explosive bulk strength, Fig. 12.3 shows that there are generally a wide range of explosives that may be chosen for a given blast pattern design.

Therefore, from an economical point of view, the best explosive is not always the least expensive but rather the one that achieves the lower blasting costs.

12.3 CHARGE DIAMETER

When explosives are used with detonation velocities that vary greatly with the diameter, which is the case of ANFO, the following precautions must be taken:

– With blastholes under 50 mm diameter, it is better to use slurries or cartridged dynamites, even though they are more costly.

– Between 50 and 100 mm, ANFO is adequate for bench blastings as a column charge and in inner charges increasing the density by 20% with pneumatic chargers and effective priming.

– Above 100 mm, there are no problems with ANFO, although in hard rocks it is better to design columns with selective charges and a good initiation system.

In large diameters with different mixtures of bulk explosives (ANFO, Slurries, emulsions and heavy ANFO) it is very economical to charge by mechanical means.

Lastly, the gelatin and granular cartridged explosives are still used in small diameters, but in medium type calibers they are being substituted for cartridged slurries and emulsions.

12.4 ROCK CHARACTERISTICS

The geomechanic properties of the rock mass to be blasted make up the most important group of parameters, not only for their direct influence upon the results of the blasts but for their interrelation with other design variables as well.

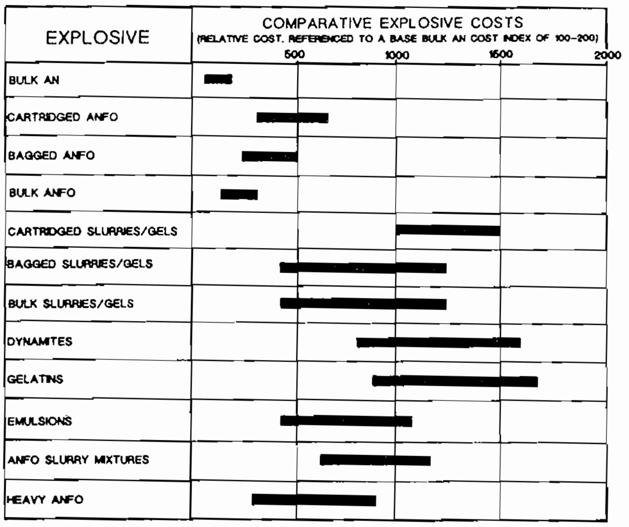


Fig. 12.1. Comparative prices of industrial prices.

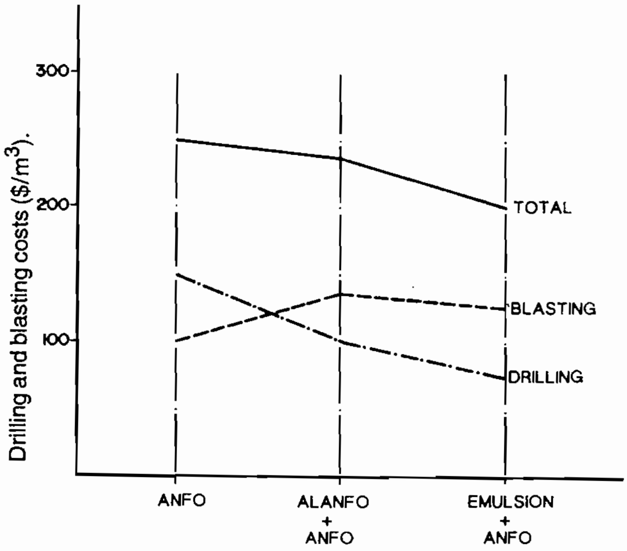


Fig. 12.2. Relative drilling and blasting costs in hard rocks for different charge alternatives.

Rocks are classified in four types, and the recommended selection criteria are:

12.4.1 *Resistant massive rocks*

In these formations, there are very few fissures and planes of weakness, which requires that the explosive create a larger number of new surfaces based upon its Strain Energy ET. The ideal explosives therefore are those with high density and detonation velocity: slurries, emulsions and gelatin explosives.

12.4.2 *Highly fissured rocks*

Explosives with high ET have very little influence upon final fragmentation in these masses because when they start to develop radial cracks these are quickly interrupted when intersected by preexisting fissures. For this reason, explosives with high Gas Energy EB, such as ANFO, are indicated.

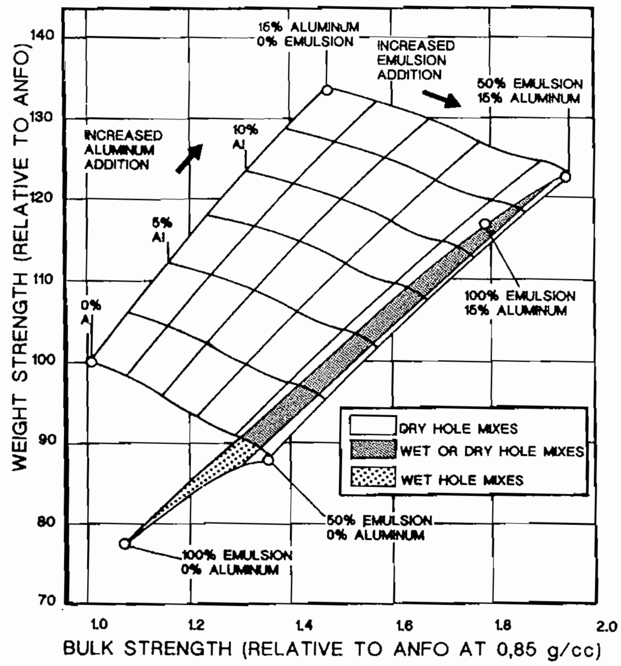


Fig. 12.3. The relationship between relative weight strength and relative bulk strength for aluminized ANFO, emulsion and heavy ANFO products indicating mixes suitable for wet and dry or dewatered hole conditions.

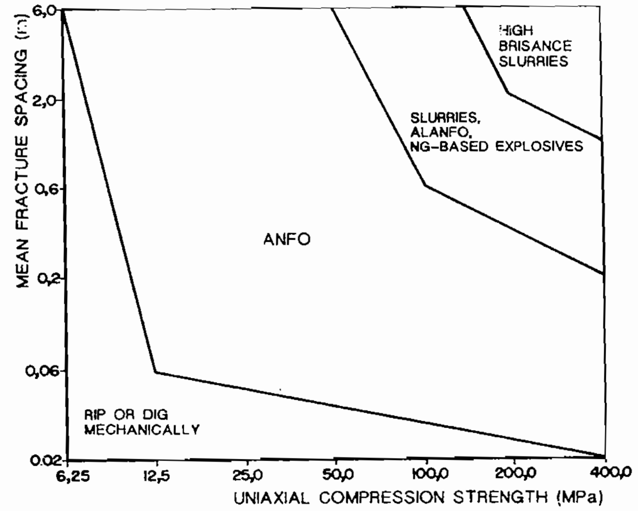


Fig. 12.4. Selection of explosives in function with the geomechanic properties of the rock masses (Brady and Brown, 1985).

12.4.3 *Rocks that form blocks*

In the masses with large spacing between discontinuities that form voluminous in-situ blocks, and in ground where large boulders exist within plastic matrixes, fragmentation is governed mainly by the geometry of the blast and, in lesser degree, by the properties of the explosive.

In these cases, explosives with a balanced ET/EB relationship are recommended, such as ALANFO and heavy ANFO.

12.4.4 *Porous rocks*

In this type of rocks, a great buffer-effect and absorption of strain energy is produced, with the bubble energy carrying out almost all the task of breakage. Apart from selecting the proper explosives, which would be those with low density and detonation velocity, such as ANFO, the following measures should be taken to retain the gases within the blastholes for as long a period as possible:

- Control the stemming material and height.
- Properly sized burden
- Bottom priming
- Reduce Blasthole Pressure by decoupling the charges or adding inert materials (ANFOPS).

12.5 VOLUME OF ROCK TO BE BLASTED

The volume of the excavation and the work schedule give the amount of explosive necessary for the breakage operation.

In large operations, the quantity of explosive may be such as to consider its use in bulk form, as it makes mechanized charging possible from the transport units themselves, reducing labor costs and making better use of the volume of rock drilled.

12.6 ATMOSPHERIC CONDITIONS

Low atmospheric temperatures have strong influence on the explosives which contain NG as they tend to freeze at temperatures under 8°C. To solve this problem, substances such as nitroglycol are used to lower the freezing point to below −20°C.

High temperatures also give problems, making the handling of explosives dangerous as in the case of so called exudation.

With the development of slurries, these risks have almost disappeared, although in cold weather the cartridges become less cap sensitive and more initiation energy is needed. ANFO is not affected by low temperatures when properly primed, but in hot atmospheres it is necessary to control the evaporation of liquid fuel.

12.7 PRESENCE OF WATER

When ANFO is in an atmosphere with more than 10% humidity, an alteration is produced that stops detonation. Therefore, when there is water in the blastholes the following should be carried out:

- If there is not much water present, pulverized ANFO is placed in waterproof plastic containers, reaching densities of 1.1 g/cm^3. Axial priming should be used because if one of the cartridges is ruptured during the charging process, it will quickly become desensitized in the column.
- If there is more water present and the former procedure is not feasible, the blasthole can be dewatered by using a down-the-hole submersible pump and then placing a waterproof liner in the hole, loading the bulk ANFO inside the liner before the water seeps in again.
- If the flow of water makes dewatering impossible, explosives such as bulk watergels and emulsions can be used, pumping or pouring them, or gelatin explosives and cartridged slurries. In this last case, the height that the water would reach can be calculated by the following formula:

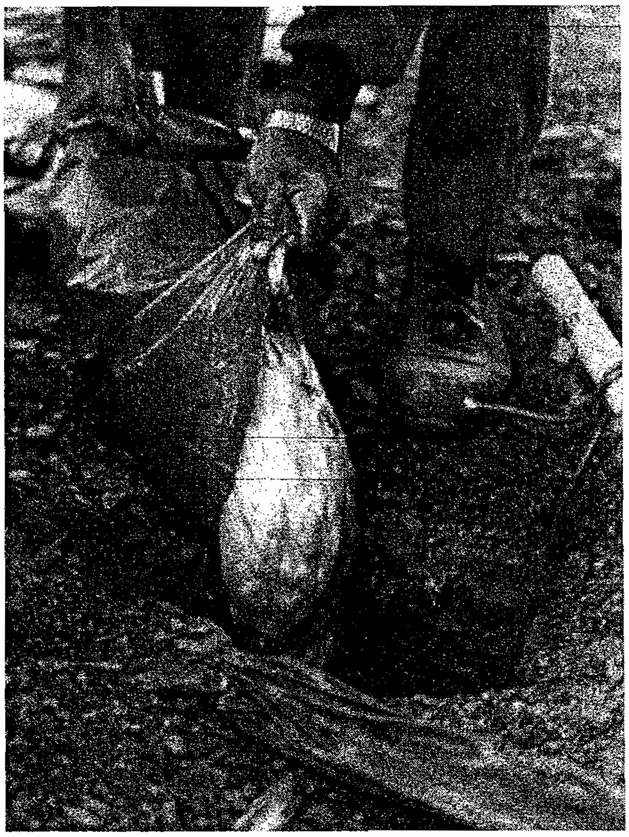


Photo 12.1. Use of pourable slurry in blastholes with water.

$$H_f = \frac{H_o \times D^2}{D^2 - d^2}$$

where: H_f = Final height of the water, H_o = Initial height of the water, D = Blasthole diameter, d = Cartridge diameter.

In cases where there is little water or dewatering is possible, Heavy ANFO has opened new perspectives for lowering blasting costs.

12.8 ENVIRONMENTAL PROBLEMS

The main problems that affect areas near the blastings are vibrations and air blast.

From the explosive point of view, those with high strain energy give higher vibration levels. Thus, when feasible, it is better to use ANFO instead of slurries. The sectioning and sequencing of charges can also be carried out with bulk and cartridge explosives by applying different initiation techniques.

As to air blast, it is recommended that the explosive have a balanced strain energy/bubble energy relationship and, above all, that the geometric design of the blasting be controlled.

12.9 FUMES

Although many explosives are prepared so that their oxygen balance gives maximum energy and minimum toxic detonation gases, the formation of harmful fumes with a certain nitrous gas and CO content is inevitable.

The fumes enter as a selection criterium in underground operations only and it must be pointed out that it is more a problem of poor ventilation rather than of the explosive itself.

The presence of plastic containers, inadequate charge diameters or inefficient initiations can bring about a large volume of fumes.

Cap sensitive slurries generally give gases with good properties, whereas with bulk slurries certain precautions must be taken, as well as with ANFO which produces a high concentration of nitrous gases.

The gelatin explosives are usually good, but not so those dynamites with a high AN content.

12.10 SAFETY CONDITIONS

A balance between sensitivity and safety is not always easy to achieve. Gelatin explosives have high sensitivity but if explosive left-overs are found in the muck pile for some reason (cut-offs, detonating cord breakage, etc.) and heavy machinery is used: front end loaders or caterpillars, detonation could occur with the consequent danger for operators. This problem has been solved with the use of slurries and emulsions that are insensitive to being struck, friction or subsonic stimulation, but have an adequate degree of sensitivity for initiation.

12.11 EXPLOSIVE ATMOSPHERES

Excavations carried out in gassy atmospheres, such as in coal mines as well as in other metal mines or civil engineering, could cause great catastrophies if secondary blasts occur.

For this reason, in such projects it is most necessary to undertake the task of studying the atmosphere and environment near the blasting so as to decide whether to use permissible explosives and/or inhibitors in the stemming material.

12.12 SUPPLY PROBLEMS

Another final factor to take into account is the available supply in function with the location of the work and the proximity of explosives and their accessories.

If the powder magazines belong to the organization, length of storage must be taken into consideration along with possible variations in the explosive characteristics of some of the products.

REFERENCES

Dick, R.A. et al.: *Explosives and Blasting Procedures Manual.* US Bureau of Mines, 1983.

Lopez Jimeno, E.: Implantación de un Método de Cálculo y Diseño de Voladura en Banco. Thesis doctoral E.T.S. Ingenieros de Minas de Madrid. 1986.

Manon, J.J.: *How to select an explosive or blasting agent for a specific job.* E/MJ May, 1977.

Union Española de Explosivos: *Explosivos y Accesorios.* 1981.

Wright, K.W.: *Effective blast round design selecting the right explosive for the right job.* World Mining Equipment. March, 1986.

CHAPTER 13

Blasting accessories

13.1 INTRODUCTION

Parallel to the evolution of explosives, initiation devices have gone through important technological advances since the decade of the forties in an attempt to obtain the following objectives:

- Energetic initiation of the more modern explosives, which are much more insensitive than the clasical dynamites but also safer.
- Control over initiation times to improve fragmentation.
- Reduction of the vibration levels, air blast and flyrock produced in blasts.
- Punctual priming, either at the top or base of the blasthole, or axial priming.
- More speed and flexibility in breakage operations while maintaining a high degree of safety for personnel and installations.

At present, the firing systems for the detonators called ordinary blasting caps by means of a safety fuse, which implies high risk for the blaster and a lack of control over ignition timing with negative effects on the blasting yields and the alterations that could occur, has almost completely been substituted by safer and more trustworthy systems that can be classified in two groups:

- Electric systems, and
- Nonelectric systems.

In this chapter, the characteristics of the different initiation devices for both groups will be discussed, along with other useful elements for the correct execution of blastings.

13.2 NONELECTRIC INITIATION SYSTEMS

13.2.1 *Blasting caps initiated by low core load detonating cords*

Very low energy cords are made up of a pentrite core with a variable core load of between 0.8 and 1.5 g/m surrounded by wires and a flexible plastic cover of about 3 mm. The blasting cap placed at one of the ends of the cord is similar to the electric version, with the only difference being that the ignitor is the cord itself, and it is usually finished off by a plastic connector as shown in Fig. 13.1, with which it is joined to the lead firing cord with the highest core load.

These detonators are commercialized in several countries under different names such as Anodet, Detaline, Primadet, etc. A great advantage is that they do not initiate blasting agents such as slurries and ANFO, allowing bottom priming to be carried out.

13.2.2 *Shock tube systems or non-electric caps*

These systems consist in a thin transparent plastic tube lined by a light coating of explosive of approximately 20 mg/m and a detonator that is similar to an electric blasting cap. The speed of the shock wave within the tube is around 2000 m/s and is not sufficiently strong enough to initiate explosives that are in contact with the tube, however sensitive they may be, which also permits an effective bottom priming.

The initiation can be carried out by means of a detonation cord, an electric blasting cap, or with a special pistol charged with blank cartridges.

The delay timings with which the Nonel, made in Sweden, are commercialized are 25 ms, 100 ms, 200 ms and 500 ms, giving a very wide range of delay sequences, depending upon the numbers of the series. In other countries where they are made under patent, the delay timings can differ.

When calculating the blasts, the delay caused by shock wave transmission through the tube, which is about 0.5 ms per meter of length, must be accounted for.

In order to give this system more flexibility and reduce costs, Nonel is used with a reduced length of tube in combination with a low core load detonating cord (1 g/m) and joining them with plastic connectors.

One practical problem that this type of detonator poses is the impossibility of checking the firing circuits, as only visual inspection can be carried out.

13.2.3 *Hercudet caps*

This system uses a special blasting machine that is connected to the caps by a thin plastic tube which closes the circuit. The blasting machine injects a gaseous mixture of two components, oxygen plus a fuel gas, into the circuit, and the explosion starts when the whole line is filled with the mixture. The detonation is propagated at a speed of 2400 m/s, initiating the caps along its way, but not the explosive in contact with the tubes, which means that bottom priming is also feasible, Fig. 13.3.

The caps are of conventional type, either instantaneous or with delay intervals of 50 ms for the first numbers and

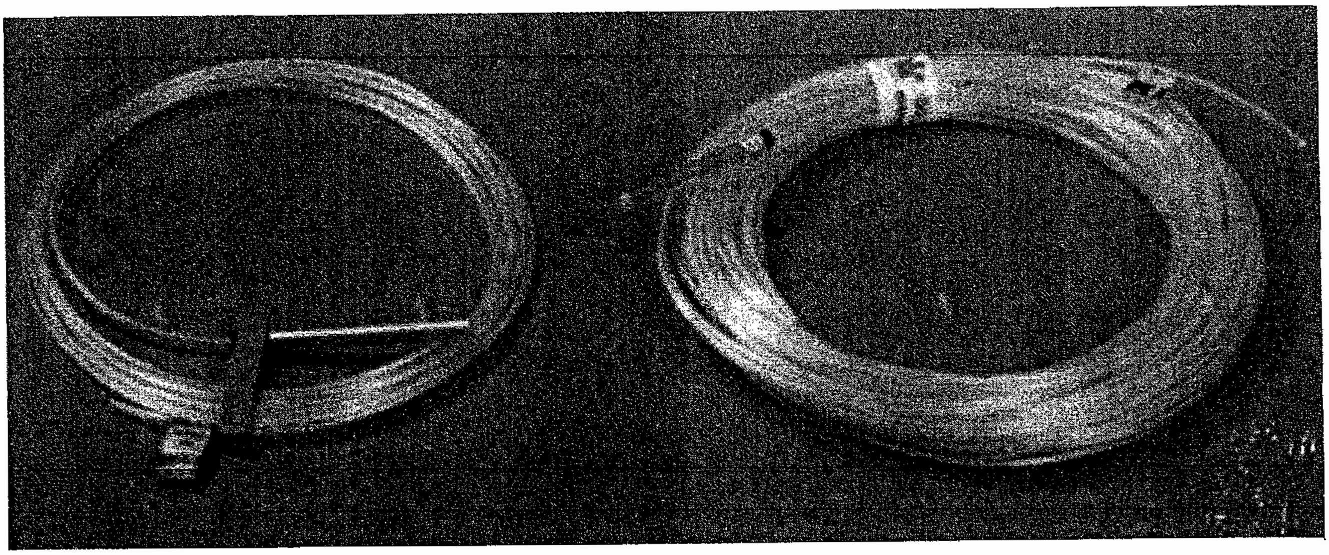


Photo 13.1. Nonel shock tubes or blasting caps (Nitro-Nobel).

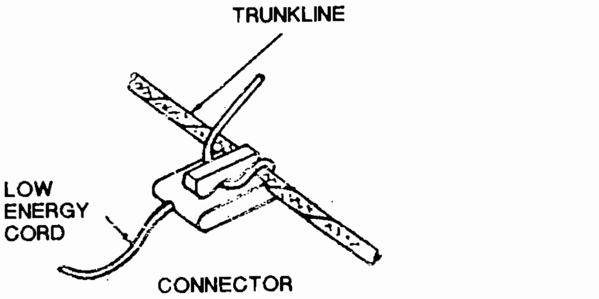


Fig. 13.1. Plastic connector in detonating cord of very low energy.

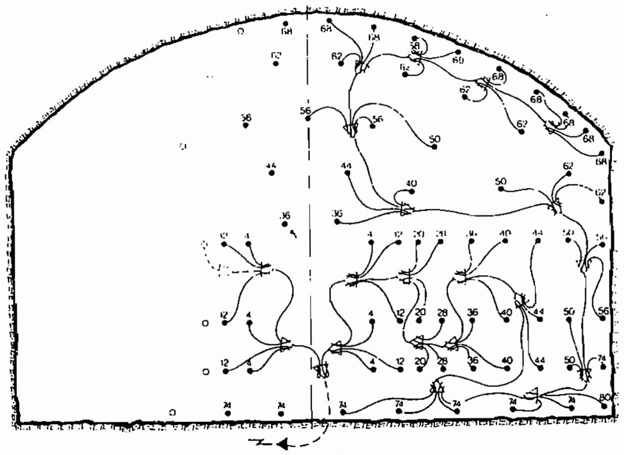


Fig. 13.2. Initiation pattern using Nonel shock tubes and connectors.

60 ms for the for the last, encompassing a total time that goes from 50 to 850 ms. The electrical part of these caps has been substituted for two plastic tubes that protrude from the cap, 10 cm for surface operations and 4.8 or 7.2 m for underground blasts.

The main advantage of this cap in comparison with other non-electrical ones is the possibility of checking to make certain that the circuit of the round has been properly constituted. A certain flow of air or nitrogen can be introduced, measuring the pressure afterwards with a pressure test module. In Fig. 13.4, a connection circuit is shown.

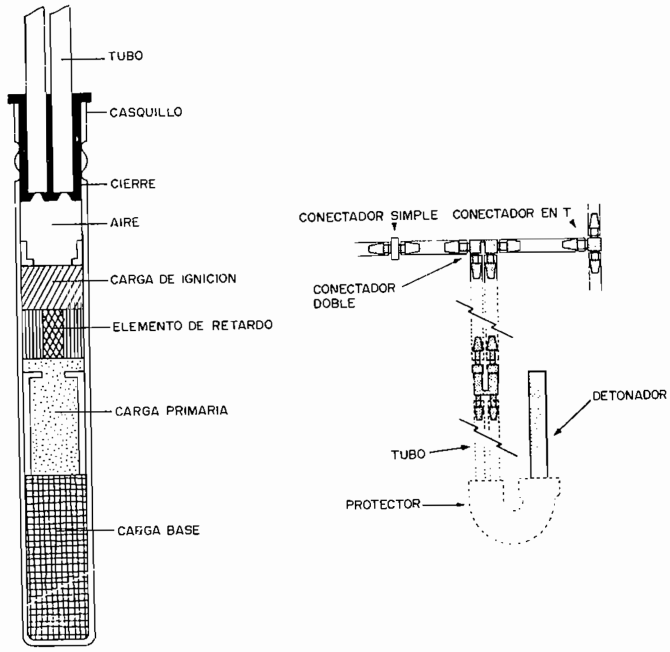


Fig. 13.3. Components of Hercudet cap and tube connectors.

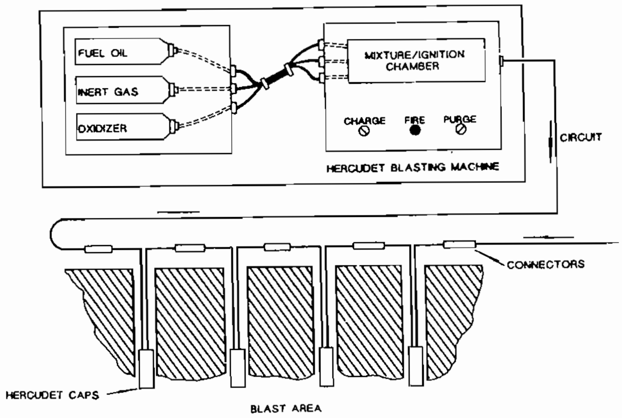


Fig. 13.4. Pattern of a circuit with Hercudet caps.

13.2.4 *Delay primers*

This group of accessories usually consists in a conventional primer with a plastic cover that has a delay channel, as a generatrix, through which the low core load detonating cord of 3 to 6 g/m passes. The *time element* is inserted in the primer and has a blasting cap or sensor near the detonating cord, a transmitter and a delay detonator.

This type of primer is used basically in blastings where deck charges and different initiation times are used in order to reduce powder factors. The nominal sequenced timings depend upon the different manufacturers, a few of which are listed below:

- Deckmaster by Atlas Powder Co. of 25 ms and 50 ms intervals with a maximum delay for the series of 400 ms.
- Austin ADP of 25, 50 and 75 ms delay intervals.
- Slider by CIL, Inc. Fig. 13.5, etc.

The number of intervals can by increased by combining this system with the usual top priming with electric milisecond delay detonators.

13.2.5 *Surface and borehole detonating cord millisecond delay connector*

Millisecond delay connectors are accessories that when connected at intervals along the trunkline of detonating cord produce delay periods for transmission of the detonation wave.

They are made up of millisecond delay connectors with two small charges encased in metal sheaths pressed against their sides.

In those of old design, the cord is connected to the metal sleeve by means of pincers, and the most modern, which are of plastic, have special holes at each end so that the detonating cord can be looped and locked in the connector by means of a tapered pin, Fig. 13.6.

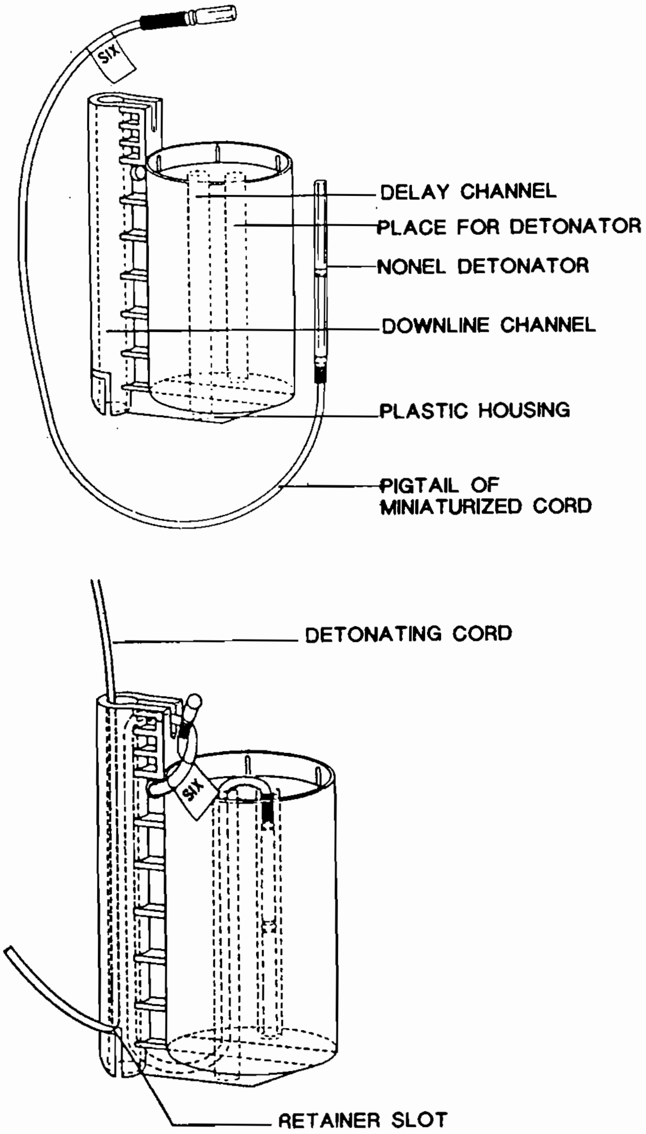


Fig. 13.5. Elements and assembly of a Slider delay primer.

Photo 13.2. Deckmaster delay primer

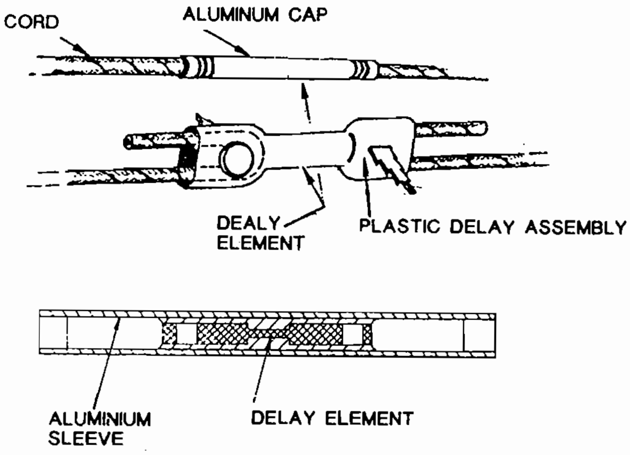


Fig. 13.6. Types of surface milisecond delay connectors.

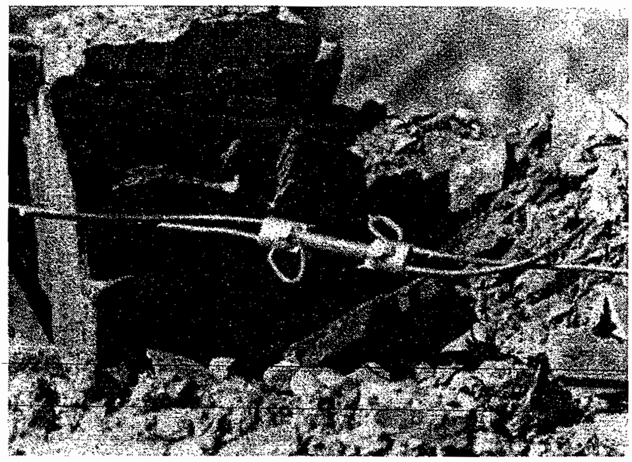


Photo 13.3. Clip-on surface detonating cord delay connector.

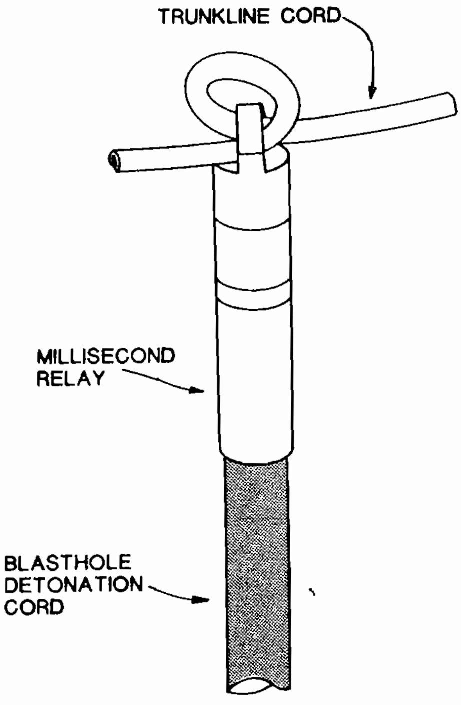


Fig. 13.7. Milisecond delay in blastholes.

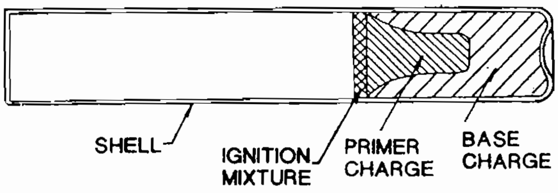


Fig. 13.8. Blasting cap to be used with safety fuse.

The delay timings are always in miliseconds and usually oscillate between 10 and 100 ms, depending upon the manufacturer. In Spain, those of 15 and 25 ms are commercialized.

With the use of these elements, sequences with an unlimited number of intervals can be achieved by placing more than one in the same strand of cord between each two blastholes.

The other delay connectors, called blasthole milisecond delays, are essentially the same as they are made up of a small aluminum cylinder with one end open where the detonating cord which initiates the charge is inserted, and at the other end there is a small ring through which the lower core load cord, that constitutes the principal line, is threaded, Fig. 13.7.

The milisecond delays vary from 25 to 1000 ms, and one is needed for each blasthole.

In order to avoid misfires, it is recommended that each blast have a double initiation circuit.

13.2.6 *Cap and fuse*

Nonelectric blasting or common caps are made up of an aluminum shell containing two types of charges: a base charge with a high detonation velocity explosive in the bottom of the shell and an igniting charge with a more sensitive explosive, Fig. 13.8.

They are initiated by a strand of safety fuse that is inserted into the common cap and crimped with a crimping tool to prevent the fuse from slipping out of the cap and to prevent water or other foreign material from entering the cap and inhibiting detonation. The safety fuse is manufactured with a core of black powder, wrapped in various layers of textile fibers and waterproofing material, that protect it from abrasion, humidity and mechanical forces.

The safety fuse should be cut squarely so that it comes well into contact with the igniting charge of the cap, Fig. 13.9.

The burning rate is usually 2 minutes per meter, with a tolerance of more or less 10%.

The use of these accessories has decreased over the years. Nowadays they are the hardly used at all.

A very useful system for simultaneously igniting a large number of fuses safely and quickly is by igniter cord, which can reach propagation velocities of up to 3 m/s, and fuse connectors, Fig. 13.10.

The lighting of the igniter cord can be done in three different ways: with the flame of a lighter, electrically or with a safety fuse.

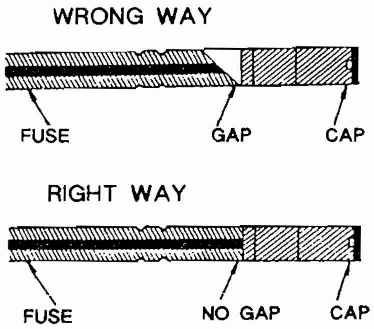
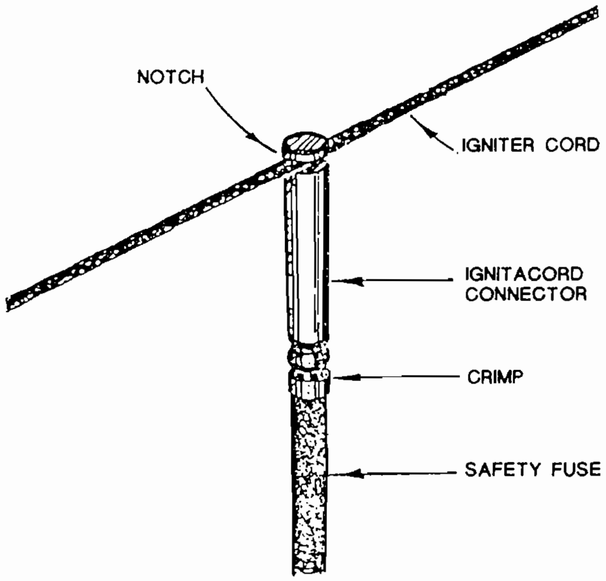


Fig. 13.9. The principle of safety fuse – blasting cap assembly.



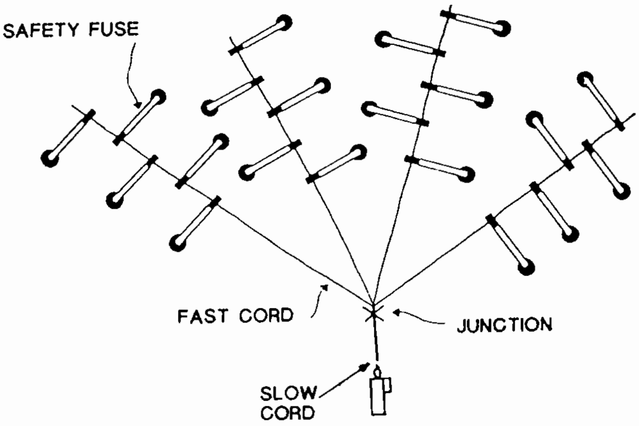


Fig. 13.10. System of rapid ignition for the safety fuse.

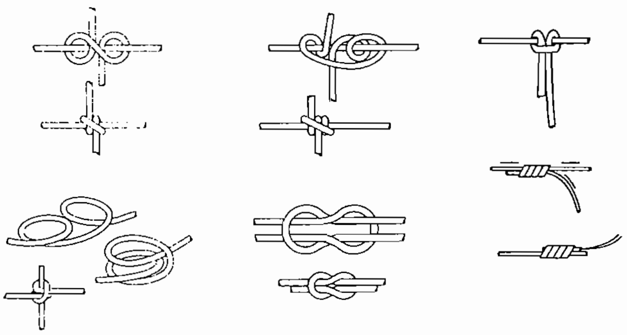


Fig. 13.11. Recommended knots for detonating cord.

Table 13.1.

Detonating cord (g/m)	Applications
1.5-3	Initiation of primers and very sensitive explosives
6	Trunklines connecting blastholes
12-20	Initiation of conventional and low sensitivity explosives
40	Seismic exploration
100	Contour blasts and demolitions

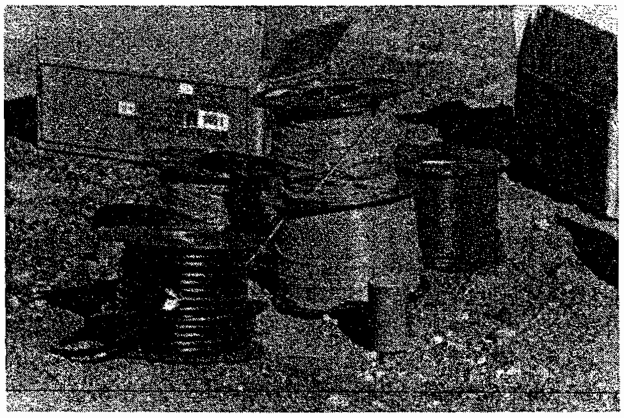


Photo 13.4. Different core load detonating cords.

13.2.7 *Detonating cords*

These cords have a central core of PETN in varying amounts (3, 6, 12, 40 and 100 g/m) wrapped in a braid of various textile fibers, and covered with a plastic jacket that gives adequate flexibility, waterproofing, tensile strength and resistance to humidity.

The detonation velocity is around 7000 m/s. The types of knots that can be made are shown in Fig. 13.11.

In 3 g/m cord such knots should be eliminated unless they are carried out with higher core load cords.

Although the primary application of these accessories is to propagate the detonation, initiated by a detonator, to an explosive charge, they also have other uses as explained in Table 13.1.

There are also reinforced cords on the market for underwater operations and other special ones that are anti-firedamp.

13.3 ELECTRIC INITIATION SYSTEMS

13.3.1 *Electric blasting caps*

These devices are made up of an aluminum or copper shell that holds an ignition system, a primer charge and a base charge, Fig. 13.12. The strength of the blasting caps depends upon the quantity of mercury fulminate in its composition, normally between 1 and 2 grams which correspond to the cap strength numbers 6 and 8 respectively, or any other equivalent explosive such as pressed PETN, etc.

If the electric blasting cap is of delay or milisecond delay type, there is a pyrotechnic delay element between the ignition system and the primer charge.

The electric blasting caps are classified in function with the following characteristics:

1. Detonation timings.
2. Electrical characteristics and,
3. Applications.

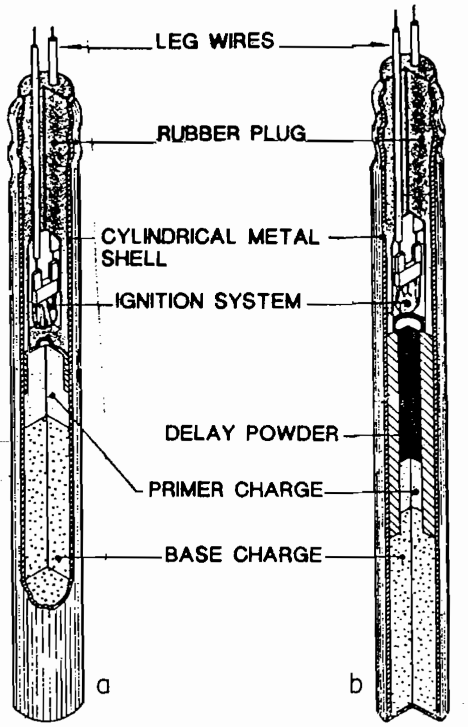


Fig. 13.12. Electric detonators. Instantaneous and delay.

Depending upon the lapse of time that passes between the moment in which the detonator is energized and the instant in which the caps detonate, they are grouped as:

- *Instantaneous caps*
- *Delay caps.* Standard delay or millisecond delay.

In Spain, the standard delay cap series (500 ms) made by UEE consists of 12 numbers, and the millisecond delay series of 20 and 30 ms with 15 and 18 numbers, respectively.

All the electric blasting caps have a certain dispersion in the initiation timings, more so in the higher numbers of the series, as demonstrated by Winzer (1979) in an exhaustive study with ultraspeed cameras. The range of variation is usually between 5 and 10 %. Birch (1983) gives an equation to estimate the standard deviation in miliseconds starting from the cap number n_d:

$$\sigma_{ts} = (3 + 2 \cdot n_d)$$

From the electrical point of view, detonators are classified according to the ignition impulse or energy per unit of electrical resistance which is needed to provoke ignition of the electric detonator. In this manner, the detonators are said to be Sensitive (S), Insensitive (I), and Highly Insensitive (AI).

The electrical characteristics of the Spanish detonators are given in Table 13.2.

As to applications, aside from the conventional, there are detonators on the market that are resistant to high water pressures for underwater blasts, copper detonators for firedamp or inflammable atmospheres and detonators for seismic prospecting.

In blastings, the electric detonators are connected to each other forming a circuit that is connected to the source of energy by the lead wire, or shooting line. There are three types of electric detonator circuits:

Table 13.2.

Electrical characteristics of the UEE blasting caps	Type of cap S	I	AI
Bridgewire resistance, Ohms (Ω)	1.2-1.6	0.4-0.5	0.03-0.05
Ignition impulse (mW.seg/Ω)	0.8-3.0	8-16	1100-2500
Safety current, Amperes (A)	0.18	0.45	4
Ignition current, Amperes (A)	1.2	2.5	25

Source: Union Española de Explosivos, S.A.

Series circuit, Fig. 13.13

The total resistance of the resulting circuit R_T is:

$$R_T = R_L + n(R_p + 2m \times r_l) = R_L + n \times R_D$$

where: R_L = Firing line resistance (Ω), R_p = Bridgewire resistance (Ω), n = Number of detonators, m = Length of the connecting wires (m), r_l = Resistance per lineal meter of connecting wire. For copper of 0.6 mm in diameter, the value is 0.065 Ω/m, R_D = Total resistance of the detonator (Ω).

This circuit is one of the most used for its simplicity and for the possibility of checking it out by simple continuation, as the current from the power source has only one path to follow. If the number of caps is high, more current is required from the blasting machine and the resulting amperage is low as it is given by:

$$I = V/R_T$$

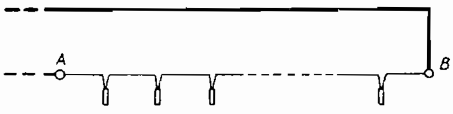


Fig. 13.13. Series circuit

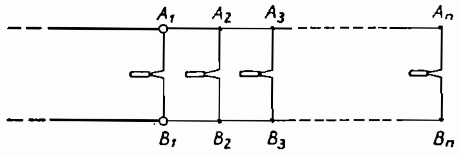


Fig. 13.14. Parallel circuit.

Parallel circuit, Fig. 13.14.

$$R_T = R_L + \frac{R_D}{n}$$

This connection system is used mostly in underground operations and is recommended when there is high risk of current leakage.

Parallel series, Fig. 13.15.

In balanced connections one has:

$$R_T = R_L + \frac{R_D \cdot n_s}{n_p}$$

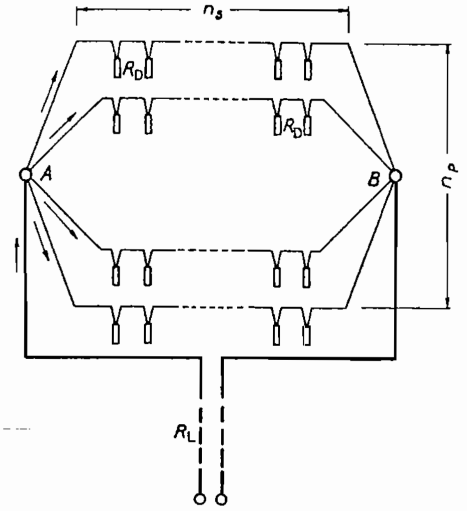


Fig. 13.15. Parallel-series circuit.

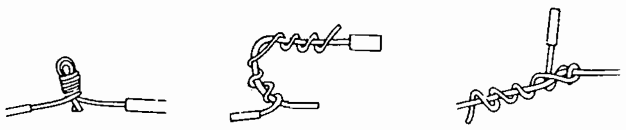


Fig. 13.16. Recommended wire splices.

where: n_s = Number of caps in a series, n_p = Number of parallel series.

This type of circuit is used when there is a very large number of caps and it is necessary to reduce the total resistance in order to adapt it to the capacity of the blasting machine.

An equation to determine the optimum number of parallel series from a given group of detonators, with the blasting machine on hand, is:

> n_p^2 = Total resistance of all electric detonators ÷ Total resistance of all firing line, connecting wire and buswire.

If, in spite of this calculation, the estimated electrical intensity is insufficient to adequately ignite the detonators, the alternatives are: change the firing line for a heavier gage with less electrical resistance, or substitute the blasting machine for one with higher voltage.

This type of circuit is very effective when the blasts have under 300 blastholes. The admissible unbalance between series is of more or less 5 percent.

The connection of the cap wires between themselves or to the buswire should be done in accordance with the recommended patterns in Fig. 13.16.

When there are no guarantees of insulation or it is desired that the connection work be carried out more quickly, speed connectors could be used.

The checking out of the circuits should be done with an ohmmeter (blasting galvanometer) that is designed so that the output intensity does not exceed 0.025 A and the short circuit current be under 0.050 A. The checking stages can be divided in the following manner:

a) *Before the circuit hookup.* Check the continuity and isolation of the firing line and, if deemed necessary, each blasting cap individually, taking the precaution of placing the cap inside a wooden block or steel tube in order to protect the blaster in case of accidental explosion.

b) *After hookup.* Check the total resistance of the circuit. When it is a series pattern, the resistances that are lower than calculated are due to a lack of connection of all the caps, or to a current leakage. If the resistance is too high, there is a false contact or the number of caps is higher than calculated. And finally, if the resistance is infinite, the circuit is open. Any one of these abnormalities can be corrected by subdividing the circuit and determining the point where the problem exists.

In the parallel circuits it is recommended that each cap be checked individually, and in the parallel-series pattern each series should be checked out and see if they are balanced.

13.3.2 *Magnadet electric detonator and Magna Primer*

In 1981, the electric detonator Magnadet appeared on the market, commercialized by ICI, giving the following advantages over conventional blasting caps:

- The stray continuous or conventional alternating currents of 50 or 60 Hz cannot initiate it.
- It passes the German static electricity test with discharges at 30 kV and 2500 picofarad, and the French test at 10 kW and 2000 picofarad.
- It is safer than the conventional caps when confronted with radio-frequency energy.
- The possibility of current leakage is practically null because each detonator behaves independently as in a parallel circuit.

The special characteristic of this detonator is that it is connected to the blasting machine through a transformer. The primary circuit is made up of the firing line which is joined to the blasting machine, and the secondary circuit of a ferrite core (toroid) and the lead wires of the each blasting cap, Fig. 13.17.

The initiation of each cap can only be produced when the primary circuit is connected to a source of alternating

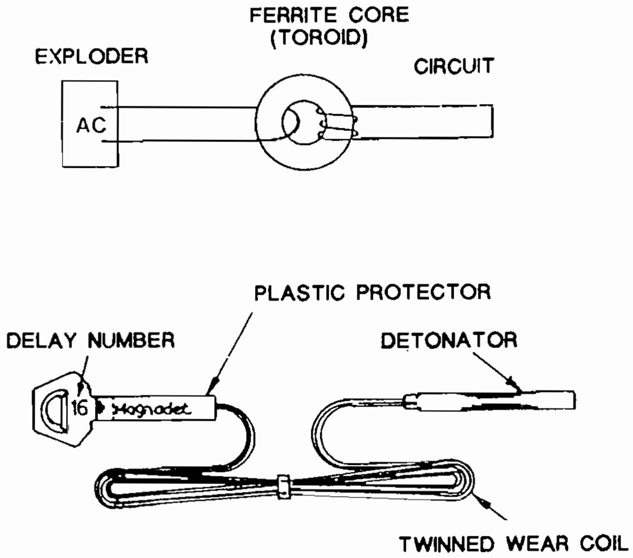


Fig. 13.17. Magnadet electric cap.

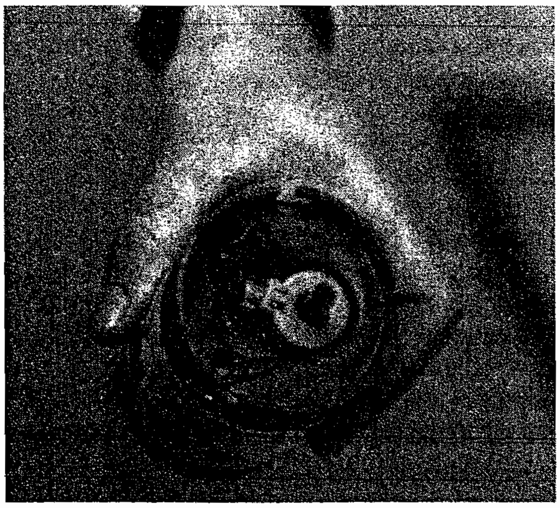


Photo 13.5. Magna Primer.

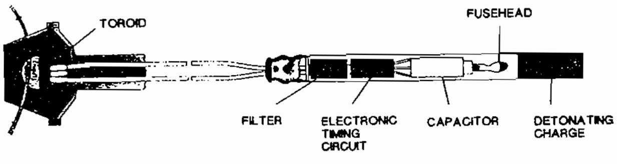


Fig. 13.18. The components of an electronic delay detonator

current with a frequency of 15 kHz or more. For this reason, special blasting machines are required that go up to 30 kHz and allow easy testing to check if the impedance of the circuit is below the acceptable limit.

The batteries are rechargeable and give up to 100 shots if maximum charging is used. The Magna primer is based upon the same principle and is designed to hold two Magnadet caps with a length of conductive wire reduced to 5 cm. The primary circuit passes through a central hole as can be observed in Photo 13.5.

13.3.3 *Electronic delay detonators*

From the middle of the eighties, several explosives manufacturers began to develop electronic delay detonators. These accessories give, due to their great precision, excellent control over the fragmentation process, as well as over vibrations and flyrock.

In Fig. 13.18, a longitudinal pattern is shown of the detonator manufactured by the ICI Group. In this device, the pyrotechnic delay detonator has an electronic timing circuit. Each detonator has a built in delay sequence number that is assigned during its manufacturing. The actual delay interval is in function with the delay number of the detonator and the interval of time is selected by the blaster and established on the blasting machine.

The detonator described is used with sequenced explosives that have different phases in their circuits that go from 5 to 250 ms, in increases of 5. The filter, in combination with the toroid, gives ideal protection against parasite currents, static electricity produced by pneumatic charging of the explosives or radio-frequency signals.

The absolute precision of these detonators is extreme, around 0.2 ms, but their price is prohibitive at the moment, about 10 to 15 times that of a conventional cap.

13.4 SOURCES OF ENERGY

The sources of energy for initiation of electric detonators are: blasting machines, batteries, and the electricity network.

The batteries, as well as the electrical lines as not very adequate sources of energy for firing blasts and should only be used in special cases and with authorization.

13.4.1 *Conventional blasting machines*

Within the group of blasting machines, the most utilized are the condenser discharge, or capacitator discharge (CD). By means of a switch or a dry cell battery the condensor is charged, closing the circuit automatically or indicating by a light or dial when the capacitator is charged. The current discharge takes place within a very short time.

In order to check out the blasting machine model to see if it is appropriate for a determined operation, or for sizing the same, the following calculations should be carried out:

1. Total available energy E_o in the blasting machine

$$E_o = \frac{1}{2} CV^2$$

where C is the capacitance in farads of the machine and V the voltage in volts that the condensor reaches at the moment of firing.

2. Energy delivered to the Circuit E_o within a short period of time, usually 5 ms.

$$E_d = E_o(1 - e^{-0.01/R_T \times C})$$

3. Effective current I_{EF}) that will be supplied by the delivered energy to the circuit.

$$I_{EF} = \sqrt{\frac{Ed}{0.005 \times R_T}}$$

4. Ignition impulse S_i.

$$S_i = \frac{Ed}{R_T} = I^2_{EF} \times t$$

Example:
A blast is to be fired of 10 blastholes with AI detonators with a spool length of 3 m and a firing line that has a resistance of 5Ω. The blasting machine has $C = 200\ \mu F$

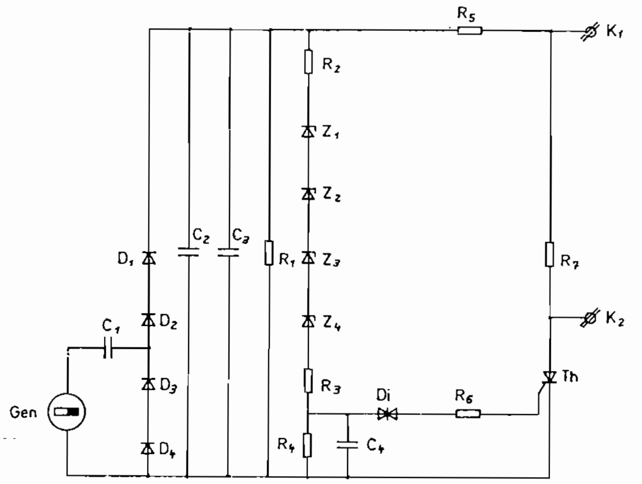


Fig. 13.19. Electric pattern of a capacitator discharge blasting machine.

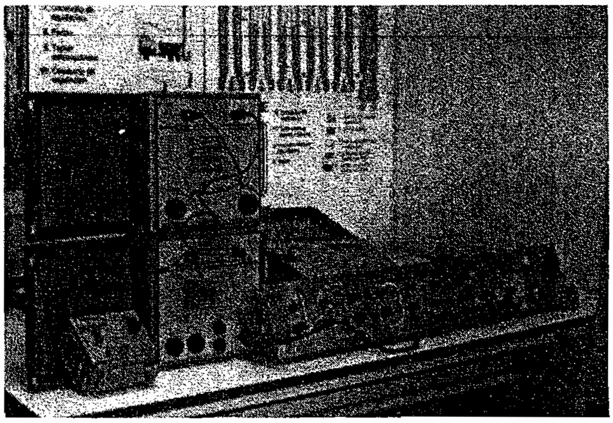


Photo 13.6. Sequential blasting unit with circuit tester, terminal board and blasting machine.

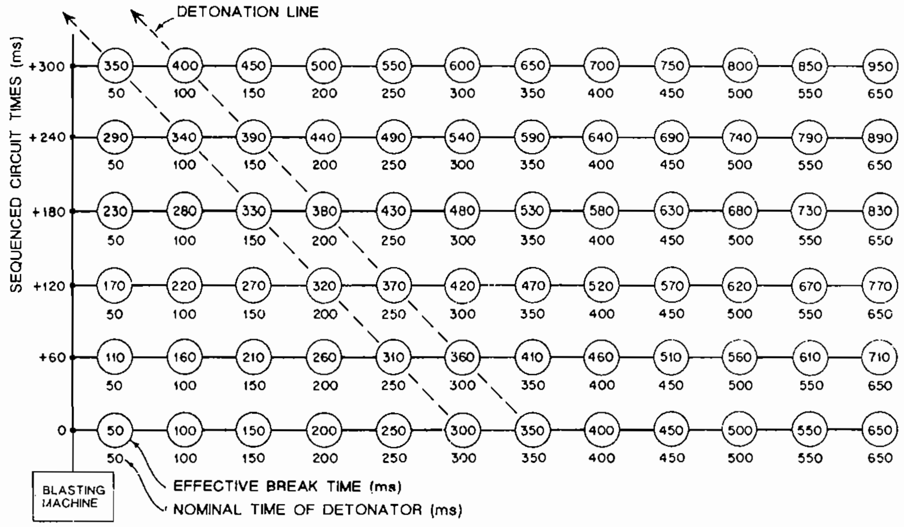


Fig. 13.20. Multiple blast fired with a sequential blasting machine.

and $V = 1.100$ V with a discharge timing of 5 ms. The blasting machine is to be checked to see if it is adequate to energize the detonators of the blast.

1. $R_T = R_L + n \times R_D$
   $= 5 + 10 \times (0.05 + 2 \times 3 \times 0.065) = 9.4\,\Omega$
2. $E_o = 0.5 \times 200 \times 10^{-6} \times 1.100^2 = 121$ joules
3. $E_d = 121 \times (1 - e^{-5.32}) = 120.4$ joules
4. $I_{EF} = \sqrt{\dfrac{120.4}{0.005 \times 9.4}} = 50.61$ A
5. $S_i = 50.62^2 \times 5 = 12{,}809.3$ mW.s/Ω

So S_i is larger than 1,100 – 2,500 mW.s/Ω which is the electric sensitivity of the AI detonators. The Safety Factor would be:

$$FS = \frac{12{,}809.3}{2{,}500} = 5.12$$

13.4.2 *Initiation by alternating current*

The energizing of blastings with alternating current from an industrial line or a generator is not recommended because, as the tension values fluctuate during a cycle of 20 ms, one never knows with what intensity the blast will be energized, which could produce misfires.

This method is usually for underground operations only.

13.4.3 *The sequential blasting machine*

In operations where the drilling diameter conditions the explosive column to be subdivided in order to reduce the operating charges, different number electric blasting caps are used in each hole.

Also, when the blasts are fired with a large group of blastholes, the normal series of electric detonators can offer a technical limitation. To avoid this problem, sequential blasting machines have recently been developed. They are basically made up of a system of discharges by condensers and an electronic delay timer that energizes various circuits at selected intervals. The most com-

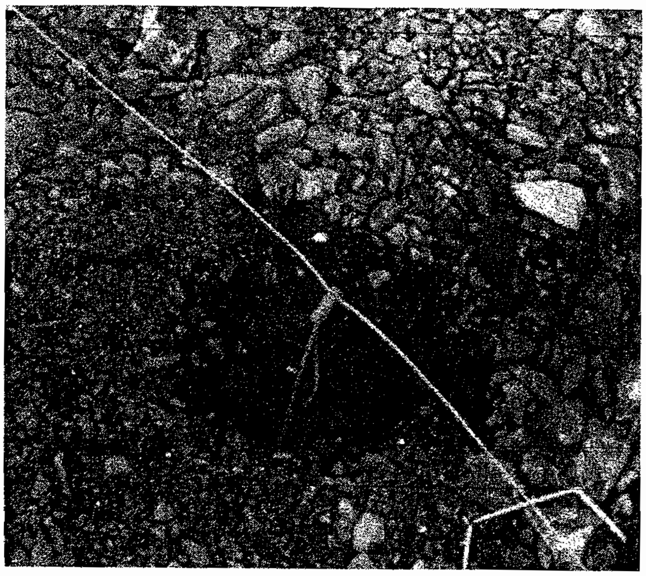


Photo 13.7.

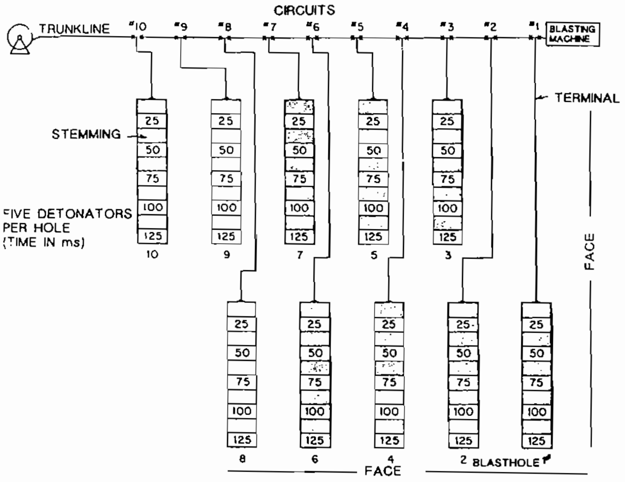


Fig. 13.21. Multideck firing sequentially.

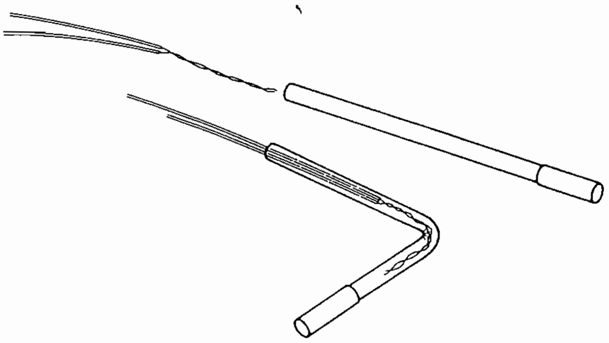


Fig. 13.22. Plastic insulating caps.

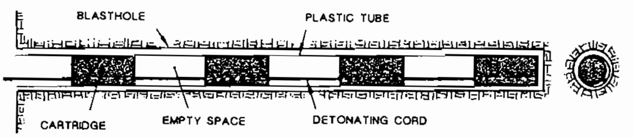


Fig. 13.23. Omega tube.

mon number of circuits is 10 and each of them can be programmed, in the most complete models, to fire one after another with increments of 1 ms between 5 and 999 ms.

The equipment is composed of the following elements: blasting machine, master cable with 10 independent circuits, machine tester and circuit tester.

The master cable, of varying length, has a core of 12 wires and 10 terminal pairs to which the ends of the circuits are connected. The blasting machine tester measures the percentage of energy that the machine is capable of supplying by discharge, and with the circuit tester, blasting galvanometer, the resistance of the different circuits is checked, according to the capacity of the blasting machine, Photo 13.6.

In Fig. 13.20, a multiple blast is represented in which a series of detonators of 12 numbers obtains 72 different outputs with 6 circuits.

Some models give remote control and can be connected to satellite equipment which is very useful in underground blasting.

13.5 OTHER ACCESSORIES

13.5.1 *Connectors*

The connectors can be of two types, depending upon their use, which may be to connect electric blasting caps or for detonating cord. The first are made up of a small tube that is closed on one end, into which the joined ends of the wires are introduced. Once this is done, the tube is bent to assure their retention, thus constituting an efficient insulation, Fig. 13.22.

The connectors for detonating cord are small plastic tubes that have a V-shaped notch on one end finishing in a diametrically placed perforation. They permit rapid and safe connection of the lines of either equal or different core load detonating cord, Photo 13.7.

13.5.2 *Omega tube and cartridges with special jointing sleeves*

In order to space the charge along the length of the blasthole in contour blasting, plastic tubes opened lengthwise are used, in which detonating cord and cartridges, separated from each other at a set distance, are placed, Fig. 13.23.

In contour blasts, the explosive can be prepared in special cartridges which have jointing sleeves at each end allowing rapid and safe preparation of the columns at the desired length.

13.5.3 *Centralizing and retention elements*

In contour blasts of small diameter where the charges are decoupled, plastic pieces shaped like a daisy and placed in tubes or cartridges are used to center these in the blastholes, leaving a coaxial ring of air that acts as a cushion, Fig. 13.24.

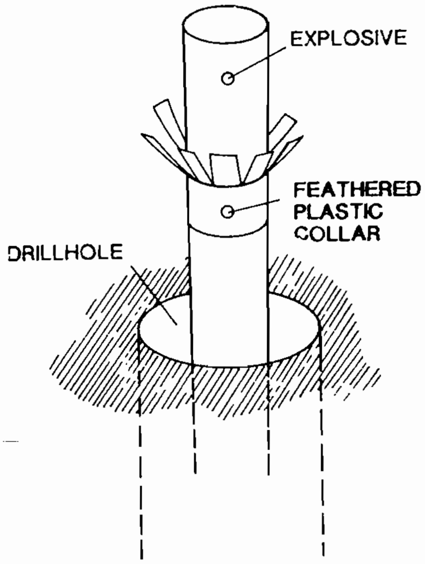


Fig. 13.24. Centering a pre-split charge in a blasthole.

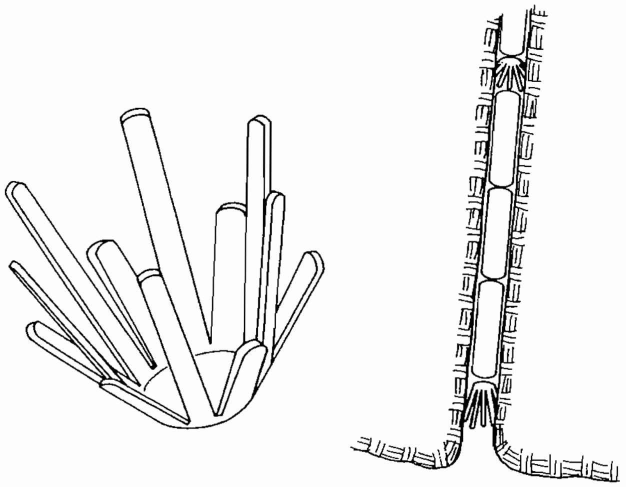


Fig. 13.25. Plastic locking springs.

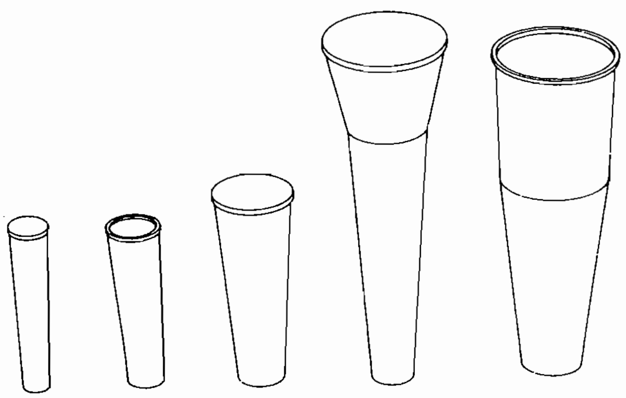


Fig. 13.26. Plastic drill hole plugs.

In underground upholes, pieces similar to those described, made of semi-rigid plastic, are used to hold the cartridges in place, Fig. 13.25.

13.5.4 *Plugs*

Plastic or wooden drill hole plugs are used in surface blastings, for marking out and for avoiding falling rocks or other objects in the drilled blastholes, Fig. 13.26.

The color of the plugs, which should contrast with the rock, can be the same throughout the blast, or the combination or various colors to visualize the pattern, not only in its geometry but in the forseen sequence as well.

13.5.5 *Funnels*

When the explosive is used in bulk form and is poured into the blastholes directly from the bags, it is convenient to have a funnel on hand to speed up charging and avoid spilling or mixing of the explosive with drill dust. These elements are made of metal plate with an outside ring to which the detonating cord is attached in order to avoid its being drawn into the blasthole, Fig. 13.27.

13.5.6 *Stemming tubes*

To carry out the stemming operation, check the depth of the blastholes and aid in their charging, stemming tubes of wood or any other material that does not produce sparks or electric charges are used. They are usually cylindrical in shape with a smooth surface and in varying lengths, with a larger cylindrical piece at the end. Sometimes they are composed of flexible parts that can be connected or plugged into each other, giving the desired length, Fig. 13.28.

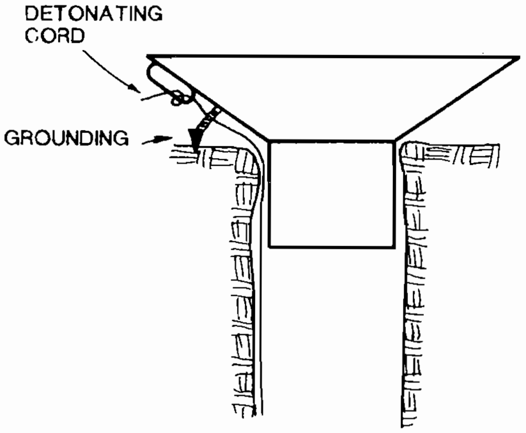


Fig. 13.27. Funnel for pouring bulk explosives.

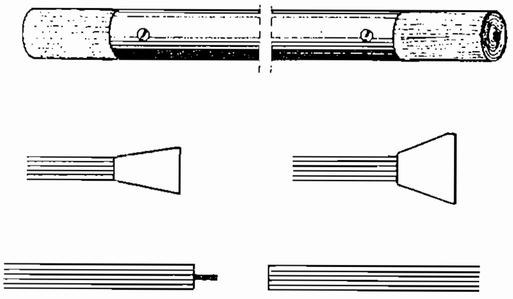


Fig. 13.28. Stemming tubes.

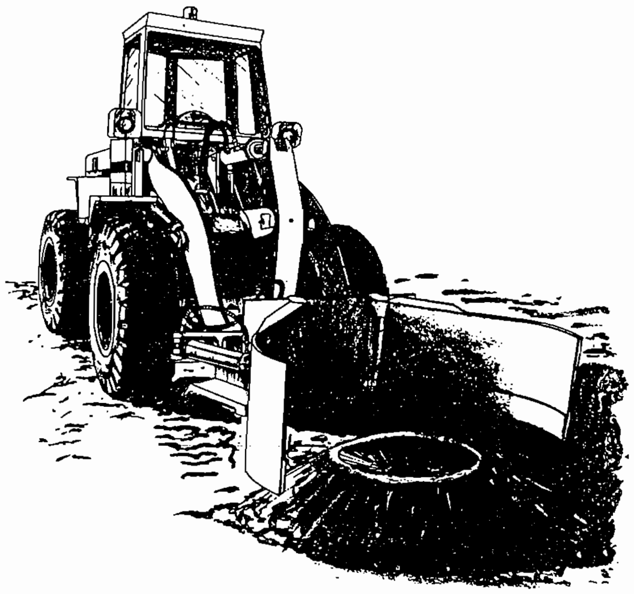


Fig. 13.29. Mechanical stemming equipment for large blastholes.

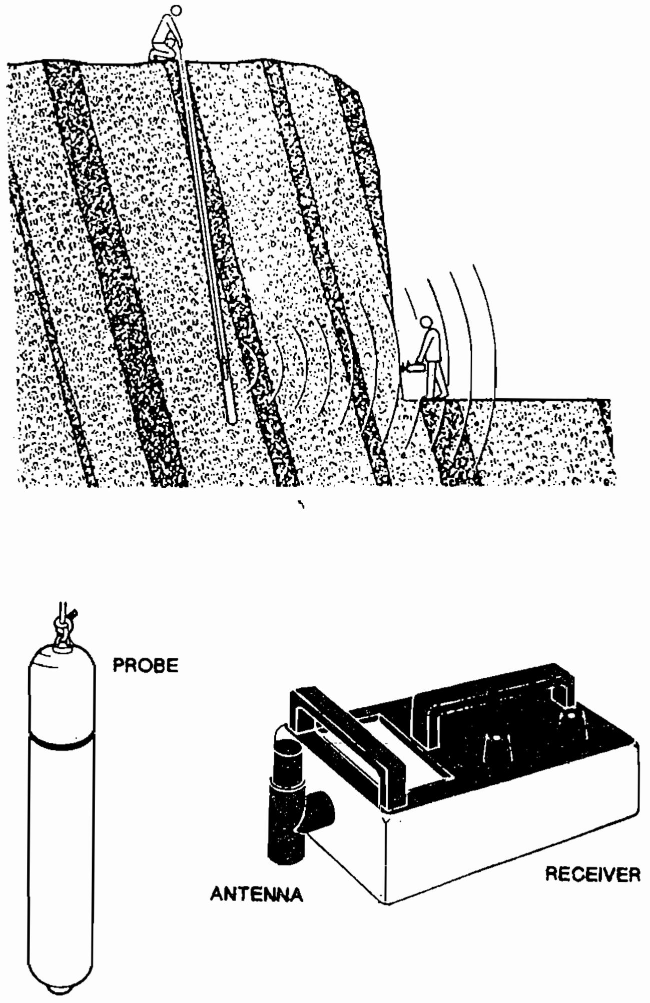


Fig. 13.30. Equipment for measuring the burden.

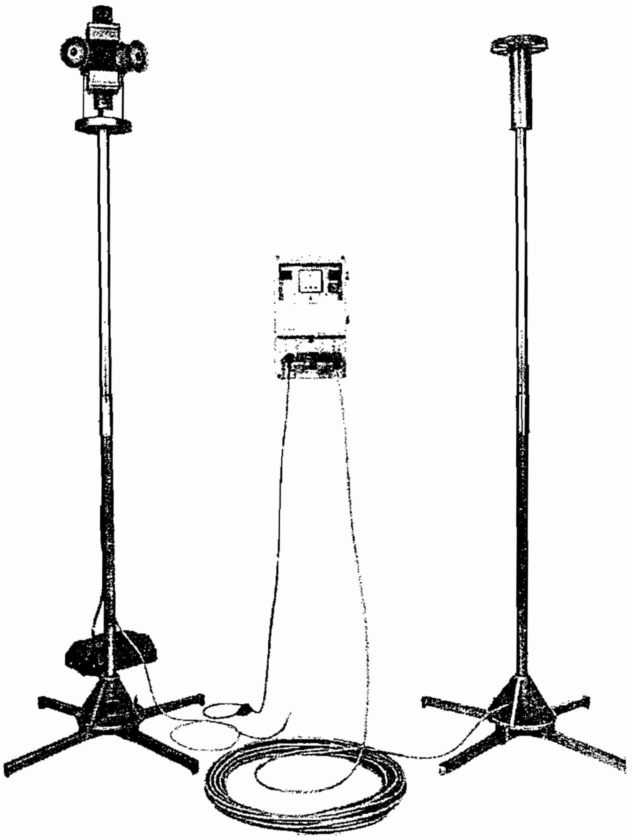


Photo 13.8. Lightning Forecasting System.

13.5.7 *Stemming equipment*

For large operations, where the number and diameter of the blastholes is so large that manual stemming becomes costly and time consuming, mechanical equipment such as shown in Fig. 13.29 came into use a several years. The machine consists of a small automotive vehicle similar to a front end.loader to which a bi-valve or pusher system, manipulated by hydraulic cylinders, has been installed.

It takes about 30 seconds to place the stemming in each blasthole and at the moment models for blastholes with diameters between 160 and 380 mm are available.

13.5.8 *Measuring instruments for burden size*

When drilling rock masses with important lithological changes and bench heights, deviations are frequent and can provoke flyrock and air blast, as well as affecting the results of the blastings.

There are instruments available to measure the burden size which consist in a probe, lowered into the blastholes, and a radiodetection aparatus that is handled by an operator from the bench toe, Fig. 13.30.

The measuring errors can be around 5%. If the deviations are not admissible, the blasthole can be redrilled or the charge pattern can be modified.

13.5.9 *Lightning Forecasting System*

The Lightning Forecasting System consists of two main parts: the sensor unit and the central unit.

The sensor unit consists of a field mill to measure electrostatic field intensity and a two-part Antenna for registration of lightning inpulse signals and atmospheric noise which is generated in thunderclouds while they develop. The antenna tube acts as a support for the field mill and is fitted to a collapsible base.

The sensor unit is located outdoors in an open place. The central unit consists of a cast aluminium cabinet with controls and signal panels, electronic equipment to evaluate the incoming signals and power supply circuits with a stand-by parallel accumulator. Furthermore, the central unit includes electronic equipment to control external alarm and recording units.

Signals fron the sensor unit are transmitted via a screened cable to the Central Unit which should preferably be located indoors or under a protective shelter outdoors.

All components subjected to corrosionn are made of stainless steel or some other non-corrosive material.

The system can be equipped with external alarm units which provide the possiblilty to distribute yellow and red alert signals throughout a working site. The units also provide a signal if the system ceases to function due to a powerfailure.

In order to be able to provide efficient warning for lightning – and high field intensity – consideration must be taken to three different types of thunderstorms as follows:

1. Thunderstorms which are already fully developed when they enter the warning zone.
2. Thunderstorms which start to develop within the warning zone. The very first lightning discharge can then occur within the zone concerned.
3. Electrostatically charged precipitation in the form of rain or snow. This type of precipitation often actuates single flashes of lightning.

Type 1 is registered through the field mill and radio wave antenna which senses the electro-magnetic radiation of the lightning flashes at a distance of about 15 km.

Types 2 and 3 are registered through the field mill which accurately senses the electrical field intensity in the air.

All types of thunderstorms are verified through registration of the radio *noise* which always occurs in developed thunderclouds.

This System provides objective alarms in two stages:

1. Yellow alert
2. Red alert

A yellow alert indicates that the electrical conditions in the atmosphere are abnormal. This can be a temporary phenomenon but, if a thunderstorm is on the way, the next stage could be a red alert.

A red alert implies that the electircal activity in the atmosphere is not temporary and therefore there is considerable risk of a flash of lightning occuring in the warning zone within 10 minutes.

Since the system can be finely set for different sensitivity, yellow and red alert levels, it can be used in many different areas which are sensitive to thunder or other forms of electrical activity in the atmosphere.

REFERENCES

Arnold Co.: *Información Técnica.*

Atlas Powder Co.: *Información Técnica. Handbook of Electric Blasting,* 1976.

Austin Powder: *Información Técnica.*

Burger, J.R.: *Nonelectric blast initiation.* E/MJ. April, 1982.

Chironis, N.P.: *New blasting machine permits custom programmed blast patterns.* Coal Age. March, 1979.

Dick R.A. et al.: *Explosives and Blasting Procedures Manual.* USBM, 1983.

Du Pont: *Información Técnica*

Garcia Milla, J.: *Optimización de Voladuras en Tajo Abierto Empleando Deckmasters y Retardos Silenciosos de Superficie Nonel Primadet.* Atlas Powder Int.

Hercules Inc.: *Información Técnica.*

ICI Explosives: *Electronic Delay Detonator.* Downline No. 7. July, 1988.

Jensen, E.B. et al.: *Nonelectric explosives detonation at the Henderson Mine.* Mining Engineering. November, 1983.

Mohanty, B. et al.: *Full-face blast rounds in shaft sinking with electronic delay detonators – A critical appraisal.* S.E.E., 1990.

Morrey, W.B.: *The Magnadet Electric Initiation System.* CIM Bulletin, November, 1982.

Nitro Nobel: *Nonel GT. Manual de Empleo.*

Norabel AB.: *Información Técnica.*

Radio Detection Co.: *GPR 110 Technical Information.*

Research Energy of Ohio Inc.: *Sequential blasting control systems and accessories.* 1984.

Rios J.: *Nuevas Tendencias en la Utilización de los Explosivos.* Canteras y Explotaciones. Abril, 1982.

Rukavina, M.: *Microchip blast cap is coming.* Rock Products, August, 1990.

Union Española de Explosivos: *Detonadores Eléctricos y Equipos Accesorios. Explosivos y Accesorios.*

CHAPTER 14

Initiation and priming systems

14.1 INTRODUCTION

The massive use of blasting agents such as ANFO, slurries and emulsions in rock breakage has brought about an important development of initiation and priming techniques. This is due to, on one hand, the relative insensitivity of these compounds and, on the other hand, a desire to obtain maximum performance from the energy released by the explosives.

The detonation process requires initiation energy so that it can develop and maintain stable conditions.

The most frequent terminology used in initiation is:

Primer. High strength, sensitive explosive used to initiate the main column in the blasthole. They are cap and detonating cord sensitive, including ones of low core load.

Booster. Powerful explosive charge with no initiation accessory that has two functions:

1. Complete the initiation work of the primer in the explosive column, and
2. Create zones of high energy release along the length of the column.

Since the seventies, various theories have been developed on initiation, some contradictory, creating some confusion amongst operators. In the following paragraphs present day knowledge is discussed and a series of practical recommendations are given in order to obtain maximum yield from the explosives.

14.2 PRIMING AND BOOSTERING BULK ANFO-TYPE BLASTING AGENTS

When blastholes have a length of under 10 m and are kept dry, initiation of ANFO can be carried out safely with only one bottom primer.

However, if the bench is very high and the holes pass through zones of different lithological characteristics and fracture frequency, water can appear and there is the possibility of separation of the explosive column during charging, due to drill cuttings and loose rock that can fall into the blasthole. In these cases, multiple priming is recommended with an initiator every 4 or 5 m, which, although slightly more costly, would eliminate the risk of incomplete detonation in any of the holes.

14.2.1 *Initiation by a primer*

In the priming of ANFO, the efficiency of a primer is defined by its detonation pressure, dimensions and shape.

The higher the detonation pressure PD, the greater its initiating ability. The effect of the 'PD' on the detonation velocity VD of ANFO is shown in Fig. 14.1.

As can be observed, with detonation pressure that is less than a certain value, a partial reduction in VD is produced, and the contrary is true when PD is above the mentioned value.

Following the same procedure, the effect of the diameter of the primer has also been studied, Fig. 14.2.

Therefore, the conditions that a primer should comply with in order to eliminate low VD zones in the ANFO are: the highest possible detonation pressure and a diameter above 2/3 that of the charge, approximately.

The length of the primer is also important, as the primer itself is initiated by a blasting cap or detonating cord and they have a run-up distance in the VD. For example, for a slurry to reach the detonation velocity regime it usually has a characteristic run-up distance of 3 to 6 times that of the charge.

In Table 14.1, the minimum dimensions of pentolite boosters for different blasthole diameters are shown.

As to the shape of the primers, the latest investigations have demonstrated that it has a significative effect upon performance, which means that it is a field open to study.

Although it is generally believed that the energy produced by ANFO increases with the transient velocity of the charge, this concept is false because the total energy released by an explosive is constant and independent of that velocity. An increase in VD brings about an increase in Strain Energy ET, thus lowering that of the gases EB but, the sum of both remains constant.

The relationship ET/EB is lower in zones of VD reduction and higher when the primer produces a raise in VD.

The increase in Strain Energy is only beneficial for fragmentation when hard, fragile and massive rocks are being blasted. In sedimentary bedding planes or highly fissured rocks, the bubble energy should be increased in order to take advantage of the fractures and planes of weakness and obtain adequate rock displacement.

Finally, it has been found that the VD steady-state in ANFO is independent of type, weight and shape of the primers (Junk, 1972).

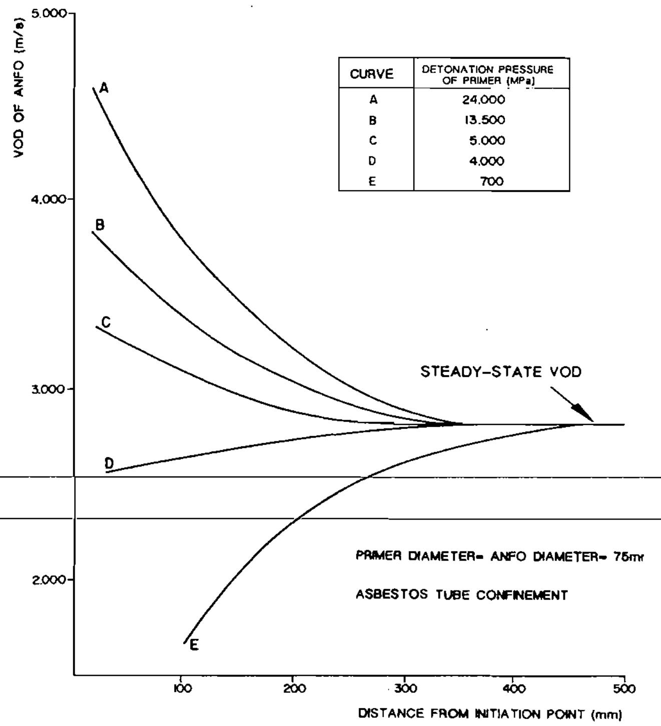


Fig. 14.1. Effect of primer's detonation pressure on initial VD of ANFO (Junk, 1972)

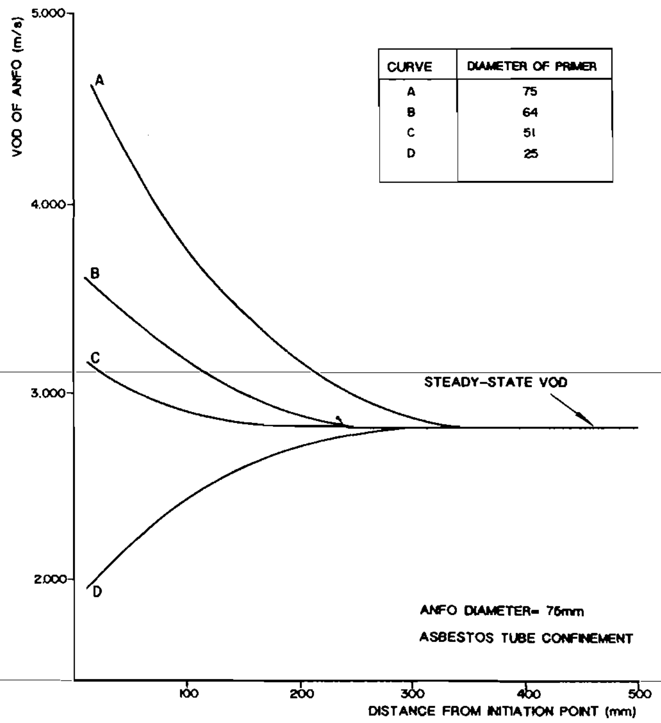


Fig. 14.2. Effect of primer's diameter on initial VD of ANFO (Junk, 1972).

Table 14.1

Blasthole diameter (mm)	Size of pentolite booster (Mass × diameter × length)
-50	30 g × 23 mm × 52 mm
50-115	60 g × 28 mm × 70 mm
115-160	150 g × 40 mm × 79 mm
160-320	400 g × 80 mm × 59 mm

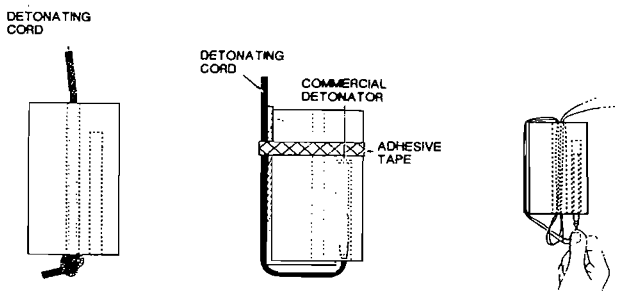


Fig. 14.3. Conventional primers.

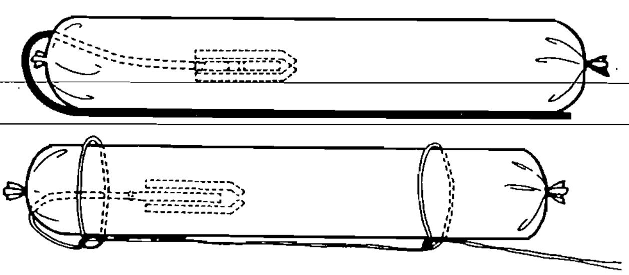


Fig. 14.4. Primer cartridges with Detaprime primer (Du Pont)

14.2.2 *Types of primers and boosters*

At present time, the most used primers are those made of pentolite as they have numerous advantages, such as:

– Insensitivity to impacts and frictions.

– High physical strength, therefore dimensionally stable.

– They have one or two longitudinal tunnels through which the detonating cord can be threaded and re-ained, or into which a detonator can be inserted, Fig. 14.3.

– They are small, compact and easy to handle, and they do not have adverse physiological effects.

– They are not alterated by age.

The slurries and emulsions that are cap or cord sensitive can be used as primers or primer cartridges, with the advantage that they occupy the entire cross-section of the blasthole and are very efficient. When these explosives require a primer for initiation, they can only be used as boosters (secondary primers) unless special accessories are used such as Detaprime by Du Pont, Fig. 14.4.

14.2.3 *Initiation by downline*

When a detonation cord has insufficient core load to initiate a charge of ANFO, the detonation of said cord creates a pressure front that expands in cylindrical shape and a chimney of gas inside the ANFO. If the cross-section of the blasthole is small then the lateral pressure can compress and desensitize the explosive.

According to Hagan, in blastholes of 75 to 125 mm, a downline with core loads of 10 g/m that lies along or near the axes densifies and desensitizes at least some of the ANFO. If the downline is along the blasthole wall, there is very little risk of desensitization with a properly mixed ANFO, but it is possible in blastholes with water where the explosive is alterated.

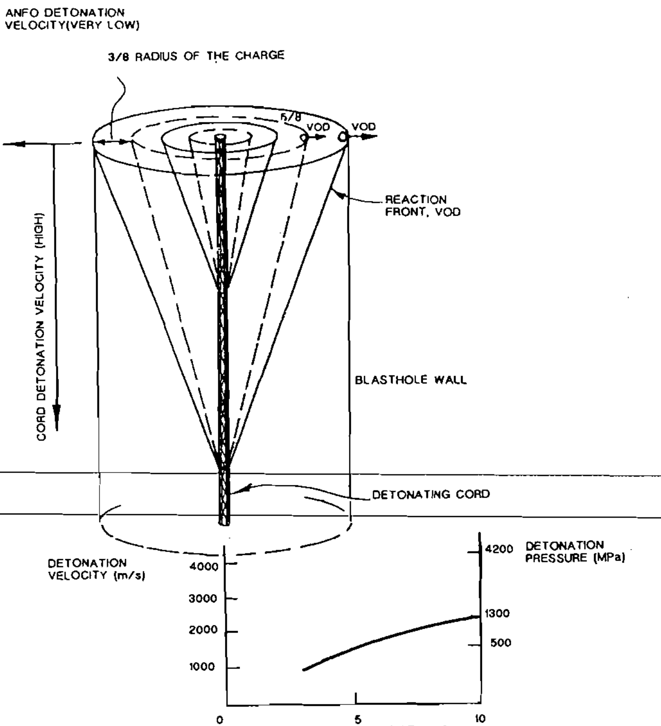


Fig. 14.5. Detonation effect of a downline lying along the axis of a blasthole upon the VD of ANFO.

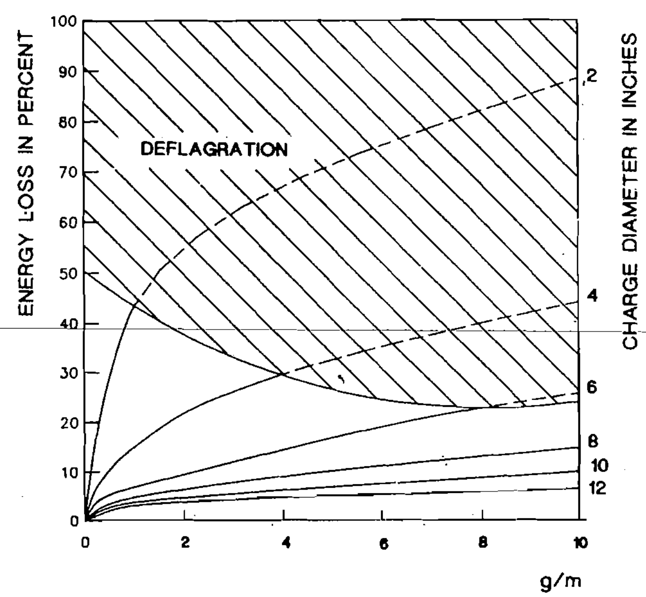


Fig. 14.6. Energy losses provoked by the downline in ANFO columns (Konya & Walter, 1990).

If the downline side-initiates the ANFO, the initial VD is slower and increases gradually while the detonation wave front passes through the section of the explosive column. With axial initiation an increase in Bubble Energy is produced at expense of the Strain Energy, which can be quite advantageous in soft and highly fissured rocks, and when a controlled trajectory blast is desired with maximum displacement.

On the other hand, in Fig. 14.6, the energy losses are shown for ANFO when it suffers damage from the downline, owing to the precompression brought on by the combustion or deflagration of part of the explosive charge.

14.2.4 *Initiation by primer and detonating cord*

When the detonating cord does not completely initiate ANFO charges, the following situations may appear:

– In blastholes with diameters larger than 200 mm and cords with core loads under 10 g/m, the detonation of the cord has an insignificant effect and the ANFO is only affected by the primer.

– When a cord of 10 g/m lies along or near the axis of a 75 to 125 mm blasthole, the detonation of the downline, as indicated before, compresses and desesitized the ANFO and prevents its detonation in parts that are not close to the primer. When this occurs, the fraction of ANFO that detonates decreases as the detonation wave propagates into the ANFO. In practice, above all in angled blastholes, as the downline lies along the blasthole wall and not the charge axis, this situation is not produced.

If the downline side-initiates the charges, the primers have little influence on the effect of the ANFO detonation, unless they are very close together.

14.3 PRIMING CARTRIDGE ANFO TYPE BLASTING AGENTS

If the covering of an ANFO charge has been damaged, permitting its contents to be alterated by water, the propagation of the detonation can be interrupted unless several primers are placed along the column of cartriged explos-

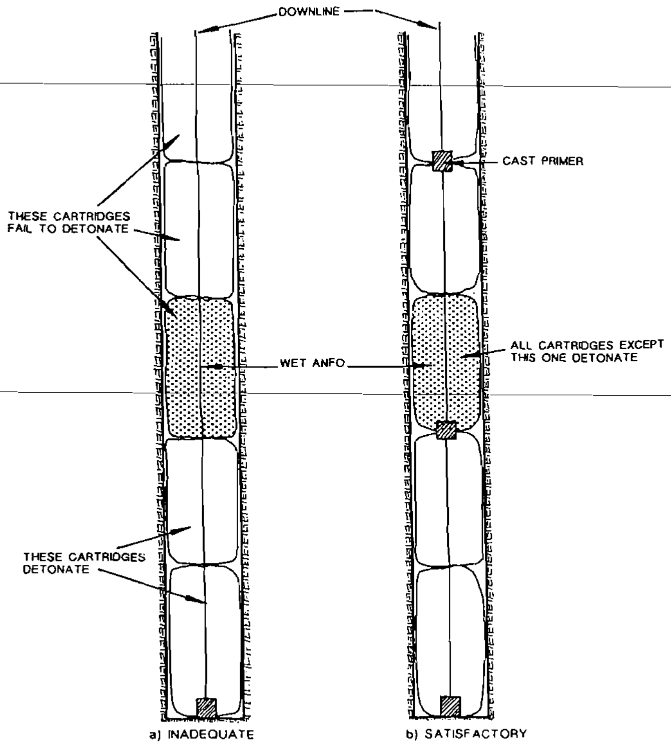

DOWNLINE
CAST PRIMER
THESE CARTRIDGES FAIL TO DETONATE
WET ANFO
ALL CARTRIDGES EXCEPT THIS ONE DETONATE
THESE CARTRIDGES DETONATE
a) INADEQUATE
b) SATISFACTORY

Fig. 14.7. Inadequate and satisfactory priming for cartridged loose-poured ANFO in wet Blastholes (Hagan, 1985).

ives, Fig. 14.7, and there is certainty that they are in contact.

In blastholes with 150 m diameters, primers of 125 g weight are recommended, and in larger holes of 500 g.

When ANFO has been pressure-packed in cartridges at the factory, the densities reached (1.1 g/cm^3) are higher than when the explosive is loose-poured (0.8 g/cm^3). Thus, although water is present in the blastholes, it is more probable that the cartridges will come into contact with the primers and, apart from this, the wrappings are usually more water and abrasion resistant, requiring less number of primers than in the previous cases.

14.4 PRIMING PUMPED OR POURED SLURRY AND EMULSION BLASTING AGENTS

Generally speaking, slurries and emulsions are less sensitive to initiation than ANFO. These blasting agents tend to be more easily compressed and can be desensitized by cord detonation inside the explosive column. Less porosity and the presence of a liquid phase reduce the atenuation of the shock wave produced by the detonating cord and prolong the action of the high pressured gases after the shock wave passes.

In order to minimize the risk of cut-offs originated by the detonating cord, in large diameter blastholes (150 to 381 mm) a multiple priming system is used. The number of equidistant boosters n_m inside a blasthole of D diameter with a column length l is determined, according to Hagan, with the following equation:

Photo 14.1. Placing a booster to initiate a column of poured slurry.

$$n_m = \frac{l}{30\,D} + 0.73$$

In a 20 m high bench with a diameter of 229 mm, a stemming of 5.70 m and a subdrilling of 1.80 m, the number of cast primers required will be:

$$n_m = \frac{20 - 5.70 + 1.80}{30 \cdot 0.229} + 0.73 = 3.07 \cong 3$$

In order to be certain that the boosters are correctly placed, a weight or heavy rock should be put on the end of the detonating cord to tense the line, and the first boosters should be placed at the calculated depth.

When the density of the multiple primers is not more than that of the blasting agents used or than that of the mud itself that can exist in the hole, there could be a risk of inadequate positioning of the primers as a consequence of their flotation or being pushed upwards. In these cases it is recommended that the downline be prepared for multiple priming outside the hole, threading twice each of the primers, Fig. 14.8.

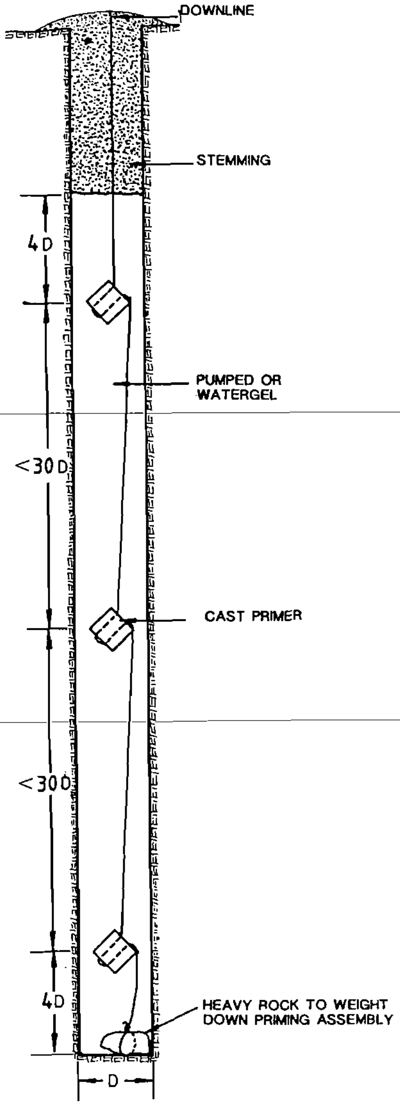


Fig. 14.8. Recommended priming system for pumped watergel and emulsion changes (Hagan, 1985).

In some place the accessories are lowered with clips in the shape of tweezers that avoid their rising towards the surface.

14.5 PRIMING CARTRIDGED WATERGEL AND EMULSION BLASTING AGENTS

Watergels and emulsions have high water resistance, which allows primers to be widely spaced within charges if it were not for the potential problem of desensitization by the downline. The multiple initiation system is recommended, as shown in Fig. 14.9. In blastholes with diameters under 150 mm, the recommended weights of the primers are 125 g, whereas in larger diameters they should be increased to 500.

As with pourable slurries and emulsions, if two lines of detonating cord are used in the blasthole, only one of these should reach the top of the column to avoid risk of desensitization.

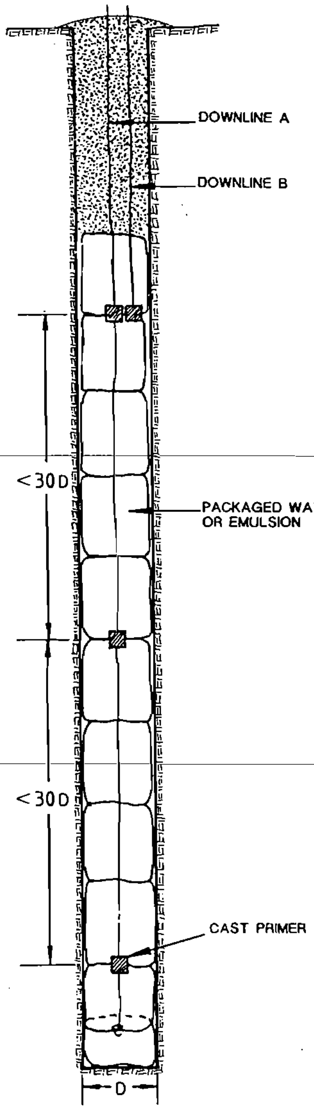


Fig. 14.9. Priming system for packaged watergel or emulsion blasting agents (Hagan, 1985).

14.6 LOCATION OF PRIMERS

14.6.1 *Bottom priming*

Bottom priming gives maximum use of explosive energy, increasing fragmentation and displacement of the rock with a minimum of flyrock. This is due to the fact that the detonation progress towards the stemming while the gases of the explosion are entirely confined within the rock mass, until the stemming material is ejected and allows their escape. This time of confinement is usually around 3 to 4 ms, according to detonation velocity and length of column. The subsequent fall of pressure through escape on bench toe level takes place much later, Fig. 14.10, achieving better fragmentation and swelling because of this, as well as a lower vibration level due to shock wave propagation towards the top part of the bench.

In bench blasts, as the breakage at floor level is extremely important, the priming should be such as to produce maximum strain at that point. If the priming takes place at floor level and not at the bottom of the blasthole, an increase in peak strain of 37% is obtained (Starfield, 1966) due to simultaneous detonation of the two parts of the charge that are equidistant from that point, Fig. 14.11. In the same manner, a 37% greater peak strain can be generated in any strong bed if the primer is placed centrally within the bed.

In blastholes without subdrilling, the bottom primer should be located as low as possible but never upon the drill cuttings or in mud, recommending that there be a distance of approximately 4D above the effective base.

Apart from the cited advantages, bottom priming has much less chance of cut-offs than top or multiple priming. In Fig. 14.12, two 270 mm diameter and 20 m long blastholes are shown as an example, where the spacing between explosive columns and stemming height is 7 m. The detonation velocities are 7000 m/s and 4000 m/s in the cord and in the ANFO, respectively, and between both blastholes there is a milisecond delay interval of 25 ms.

As blasting failures are produced by cut-off of the cord through ground movement, the larger the difference in

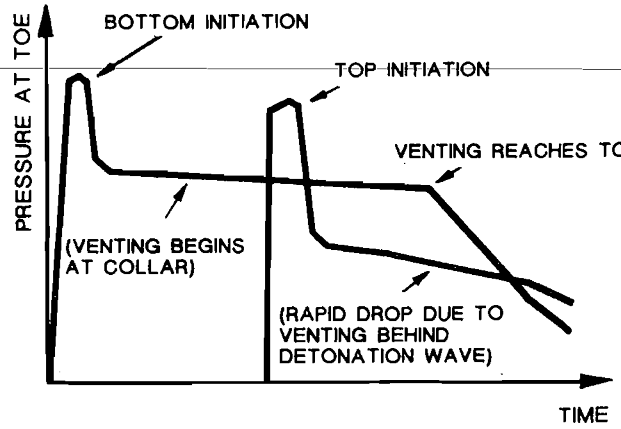


Fig. 14.10. Effect of the position of the primer upon the pressure-time profile in the blasthole.

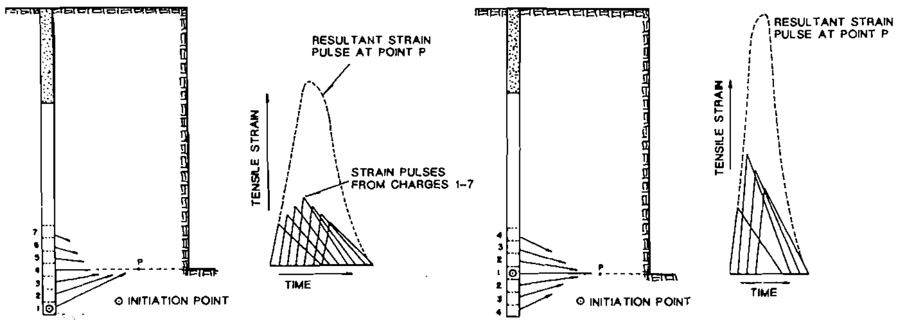

RESULTANT STRAIN PULSE AT POINT P
TENSILE STRAIN
STRAIN PULSES FROM CHARGES 1-7
TIME
7
6
5
4
3
2
1
P
⊙ INITIATION POINT
4
3
2
1
2
3
4
P
⊙ INITIATION POINT
RESULTANT STRAIN PULSE AT POINT P
TENSILE STRAIN
TIME

Fig. 14.11. Strain pulses at point *P* for charges primed (a) at their bases and (b) at bench floor level (Hagan, 1974).

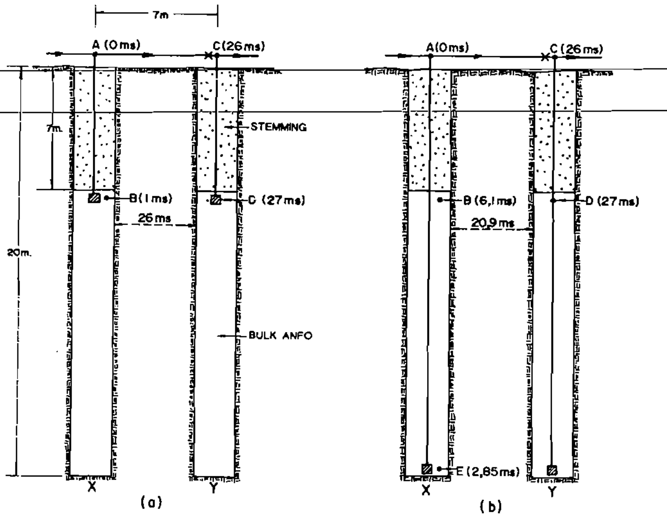


Fig. 14.12. The reduced probability of cut-offs where charges arebottom primed.

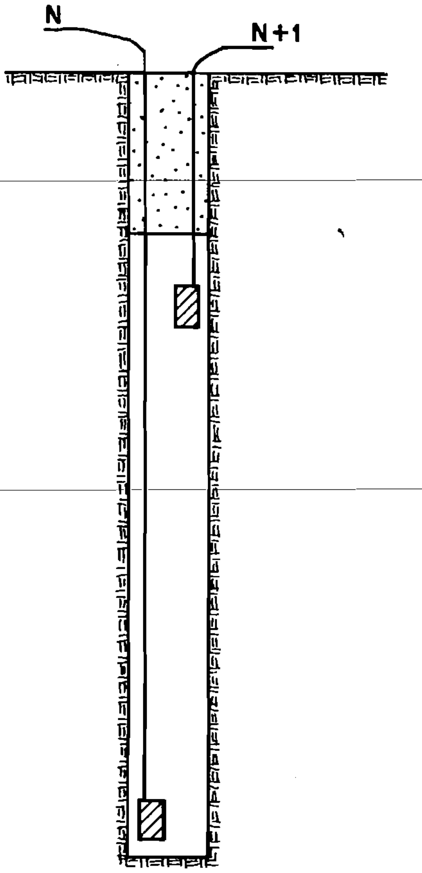


Fig. 14.13. Safety pattern with bottom priming.

detonation times in two similar points of both charges, the lower the probability of this occurring. In Fig. 14.12, it can be seen that with top priming (a) the difference in timing between B and D is 25 ms, whereas with the bottom priming (b) that time is reduced by 20%, thereby lowering the risk of cut-offs.

A bottom priming pattern called *safety* is the one indicated in Fig. 14.13.

In this case, if the low core load cord of the detonator *N* failed for some reason, at the end of a time equal to the nominal interval of the series of milisecond delay the top primer would initiate, producing the detonation of the blasthole.

Up until a short time ago, operators were not interested in bottom priming because the use of detonators inside the blastholes had certain risks, but nowadays nonelectric accessories are available such as low core load downlines and those of very low energy that offer a wide field of possibilities for this initiation system.

14.6.2 *Top priming*

In bench blasts where top priming is used, a high strain wave is propagated towards the subdrilling zone where, of course, its energy is dissipated and therefore wasted.

In blasting overburden for a dragline, this strain energy can be more usefully employed in fragmenting the rock between the bottom of the blasthole and the top of the coal, but not the coal itself, especially if there is a strong bed immediately above the coal and/or a well defined zone between the waste and the ore.

If peak strain is to be maximized along the rock that surrounds the stemming column, the top primer should be at least 1/4 of the burden below the top of the charge (Starfield, 1966). If the explosive is initiated with a primer at the highest point, the superposition of the strains generated by adjacent charge elements gives a lower result in any point of the stemming, Fig. 14.14.

The elimination of premature escape of the gases into the atmosphere, with adequate stemming height, improves fragmentation and rock displacement by Bubble Energy. For elongated charges, the efficiency of the stemming with top priming is less because the inerte stemming material, as well as the rock itself at the top, start moving some miliseconds before detonation of the lower part of the explosive. The fall of the presure of the gases is greater in long explosive columns with low

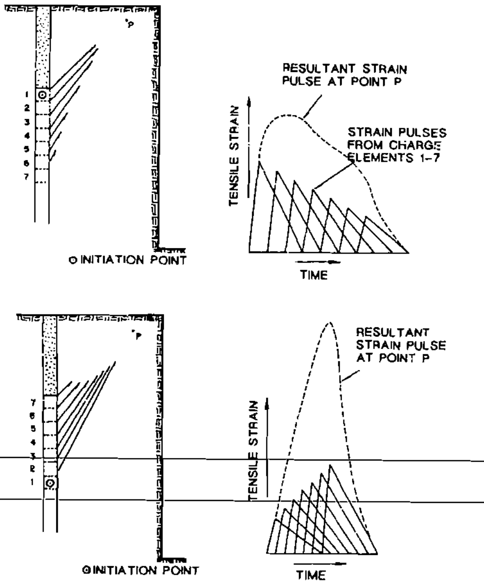

P
1
2
3
4
5
6
7
INITIATION POINT
RESULTANT STRAIN PULSE AT POINT P
STRAIN PULSES FROM CHARGE ELEMENTS 1-7
TENSILE STRAIN
TIME
P
7
6
5
4
3
2
1
INITIATION POINT
RESULTANT STRAIN PULSE AT POINT P
TENSILE STRAIN
TIME

Fig. 14.14. Strain pulses on burden alongside stemming column for charges primed at and somewhat below their uppermost point.

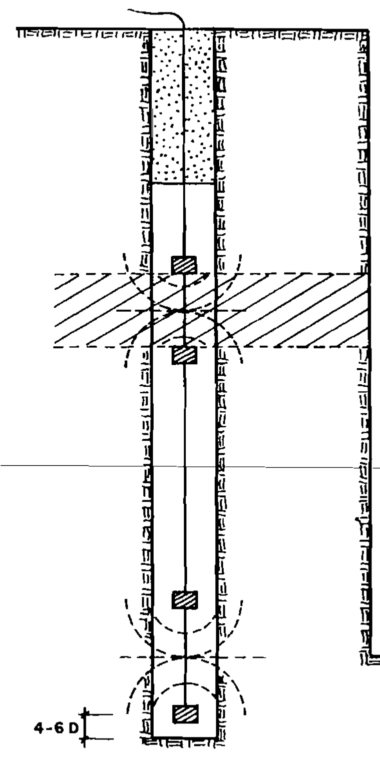


Fig. 14.15. Applications of multi-point priming in a bench blast.

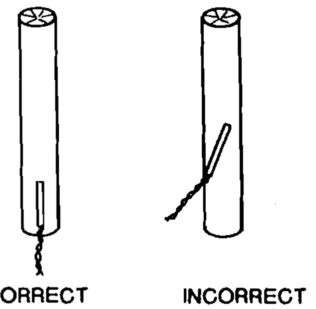


Fig. 14.16. Cartridge priming with an electric detonator.

detonation velocity and insufficient stemming, or small burden size.

When the detonation reaches bench floor level, the pressure of the gases falls rapidly from its highest value, due to their escape towards lower pressure zones. This phenomenon gives poor fragmentation in the bottom of the blasthole and especially a reduced displacement of the lower rock.

14.6.3 *Multi-point priming*

If various primers are used, they should be located in positions such as to produce collision of the detonation waves at a level that coincides with the hard zone of the rock or at bench floor level, Fig. 14.15.

The strains produced in these points of collision are 46% stronger than those obtained with simple priming (Starfield, 1966).

When the charges do not offer loss of velocity, fragmentation is improved in multi-point priming through strain energy reinforcement.

14.6.4 *Continuous side initiation*

When the explosive columns are continuously side initiated by a detonating cord (downline), the detonation velocities are relatively lower than the regime. Thus, side initiation is more effective in highly fissurized soft rock

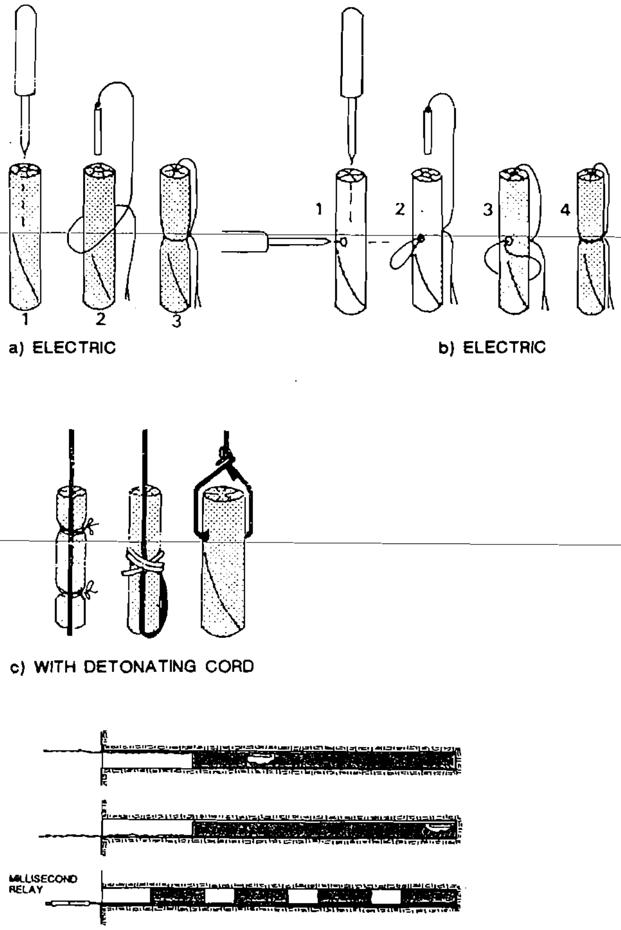


Fig. 14.17. Priming cartridges and blastholes.

formations where more bubble energy is preferible. The theory that continuous side initiation significantly increases the VOD of ANFO cannot be maintained, as has been demonstrated in practice.

14.7 PRIMING CONVENTIONAL CARTRIDGED EXPLOSIVES

The priming of cartridges consists of inserting a detonator or the end of a detonating cord in the cartridge to activate or initiate the detonation of the main charge in the blasthole.

To maximize the use of the shock effect produced by the detonator, this should be placed axially to the primer cartridge and to the axis of the explosive column, Fig. 14.16.

Any primer is an activated explosive ready to detonate under different stimulations, fire, strikes, etc., which means that they must be handled with extreme care, in transportation as well as when being placed in the blastholes. They should never be directly tamped.

For priming cartridges and blastholes with electric detonators and detonating cords, the patterns given in Fig. 14.17 should be followed.

The procedures for priming blastholes are as follows:

a) With instantaneous electric detonators. For isolated or simultaneous blastholes in rock of low to medium strength. Wet blastholes.

b) With electric delay detonator. Bottom priming for simultaneous blastholes or without a face, without water and in medium to hard type rock. With this system fragmentation is improved.

c) Detonating cord. Contour blasthole or in soft rock, with decking to lower the total charge along the length of the column.

REFERENCES

Anonymous: *Puzzled about primers for large-diameter ANFO charges? Here's some help to end the mystery.* Coal Age. August, 1976.

Anonymous: *Safe and efficient initiation of explosives.* Downline, ICI, No. 7, 10, 1988-1990.

Condon, J.L. & J.J. Snodgrass: *Effects of primer type and borehole diameter on ANFO detonation velocities.* Min. Cong. J. June, 1974.

Goffart, P.: *L'Amorcage rationnel des grosses mines charge's d'explosif envrac.* Annales des Mines de Belgique, September, 1977.

Hagan, T.N. & C. Rashleigh: *Initiating systems for underground mass firing using large diameter blastholes.* The Aus. IMM. 1978.

Hagan, T.N.: *Optimum priming systems for ammonium nitrate fuel-oil type explosives.* The Aus. IMM. July, 1974.

Hagan, T.N.: *Optimum initiating, priming and boostering systems.* AMF. 1985.

Junk, N.M.: *Overburden blasting takes on new dimensions.* Coal Age, January, 1972.

Konya, C.J.: *Initiierungstechnick für Lange Bohrlochladungen.* 1974.

Konya, C.J. & E.J. Walter: *Surface Blast Design.* Prentice Hall, 1990.

Neil, I.A. & A.C. Torrance: *The influence of primer size on explosive performance.* Explosives in Mining Workshop. The Australasian Institute of Mining and Metallurgy. 1988.

Smith, N.S.: *An investigation of the effects of explosive primer location on rock fragmentation and ground vibration.* University of Missouri-Rolla. 1980.

Thiard, R. & A. Blanchier: *Evolution des systemes d'Amorcage.* Industrie Minèrale Les Techniques. Fèvrier, 1984

CHAPTER 15

Mechanized systems for charging and dewatering blastholes

15.1 INTRODUCTION

Along with the development of explosives, charging and dewatering blastholes has been subjected to mechanization, driven by the numerous advantages that this offers to blasting as described below:

- Better use of the volume drilled in rock by being able to fill the entire blasthole with the explosive and put it into contact with the blasthole walls.
- Increase in charge density inside the blastholes.
- The possibility of forming selective charges by varying densities and specific energies along the column length.
- The use of bulk or loose-poured explosives which are less costly than cartridged.
- Less charging time.
- Less personnel required for the charging operation.
- The possibility of using ANFO, of lower cost than watergels and emulsions, after dewatering the blast-oles.
- Better control over explosives and their supply.

All these advantages lower drilling and blasting costs as the drilling patterns can be more open and the charging times reduced.

15.2 MECHANIZED BLASTHOLE CHARGING SYSTEMS

The mechanized charging systems are classified in two large groups, depending upon whether they are merely charging instruments or integral systems of manufacture and charge.

In the following, the present day methods for the most important types of explosives are described:

- Cartridged slurries and gelatin dynamites.
- ANFO and its derivatives (ALANFO and Heavy ANFO).
- Bulk slurries and emulsions.

15.2.1 *Cartridged explosives*

Pneumatic cartridge charging equipment was developed in Sweden during the decade of the fifties. These units allow the charging of blastholes with diameters between 35 and 100 mm, obtaining a 15 to 20% increase in packing densities when compared to manual tamping, or even up to 30% if a robot is used.

The charging capacities for these systems, with a two-man team, oscillates between 500 and 1.000 kilos per shift, depending upon the cartridge sizes.

In Table 15.1, the recommended diameters of cartridges and hose for different blasthole diameters are indicated.

The chargers, Fig. 15.1, consist of a tubular chamber with a flip valve at each end, a charging funnel through which the cartrigdes are introduced, a plastic loading hose and an ensemble of pressure-release pneumatic valves.

The pressurized air reaches the charger at a maximum pressure of around 1 MPa and with a series of regulators, it is reduced to 0.3 MPa. There is also a safety valve.

The loading hoses are made of black anti-static plastic, although in certain special operations metal tubes can be used. The diameters of these hoses is in function with the cartridge sizes, and its length should not exceed 50 m. At the end of the hose where the explosive emerges there are cutter blades which slit the cartridges open, and the force of ejection drives them to the bottom of the blasthole, compacting and completely filling it.

The tamping of these units is done manually, unless a *Robot*, which can be attached to the charger, is used, Photo 15.1, which substitutes the operator in this tedious and tiring work, especially in long blastholes, and allows a more regular and uniform charging.

This complement consists of a double-action pneumatic cylinder with a piston that is joined to a pneumatic pusher, a front spacer tube and a support that holds the aparatus in place against the blasthole. The cylinder has an oscillating movement that is transmitted by the pusher to the loading hose which, upon return, allows another cartridge to emerge. The degrees of stemming achieved with the forward movements of the hoses vary between 1.4 and 1.6.

The use of these chargers is especially interesting when the rounds are made up of horizontal blastholes or long, inclined upholes. The only limitations are based upon the sensitivity to impact or friction of the cartridges, thus in some instances the velocity has to be drastically reduced.

Owing to the recent tendency towards using large diameter blastholes, above 100 mm in underground mining the conventional chargers have become useless. However, the largest chargers on the market with hose centralizers have been used. This way, the cartridges of emulsion or slurry make impact in the center of the column, reducing the risk of dislodging or falling back of

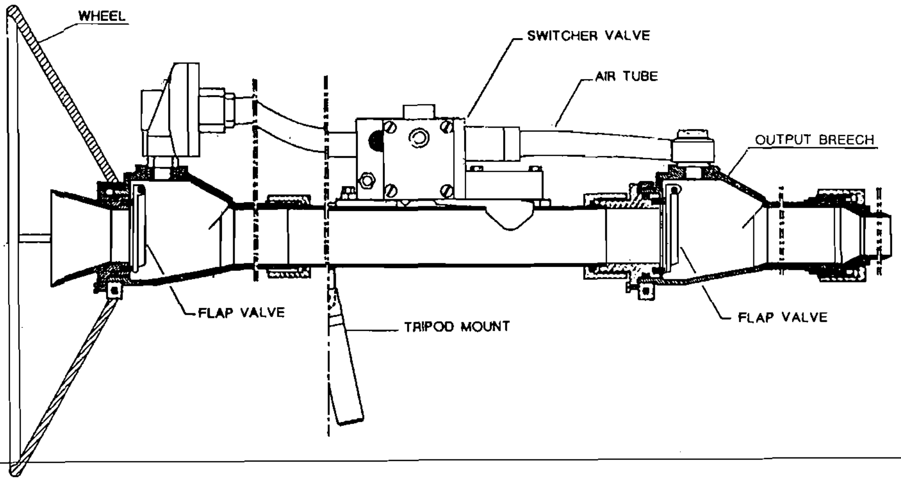


Fig. 15.1. Pneumatic charger.

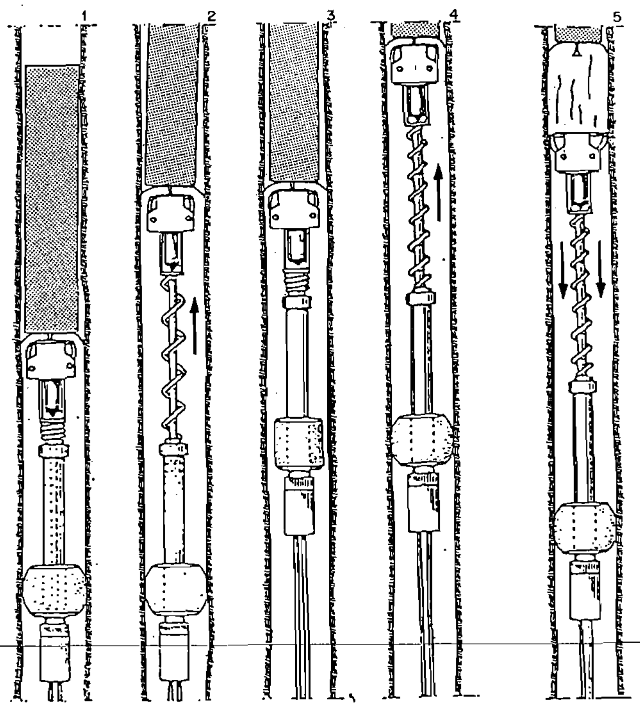


Fig. 15.2. The *Half-Pusher* technique (Nitro-Nobel).

Table 15.1.

Drill bit diameter min/max (mm)	Cartridge diameter (mm)	Hose dimensions (mm) Inside diam.	Outside diam.
38-45	22	23.2	30
40-51	25	27	34
45-64	29	30	38
51-76	32	33.5	41.5
64-102	38-40	41	51

the explosive in upholes. It has also been demonstrated, in experimental tests, that a standoff distance must be maintained between the end of the loading hose and the explosive column. The optimum is 45 cm for 165 mm blastholes, and 60 cm for those of 100 mm diameter.

In order to reduce friction of the cartridges against the inside walls of the hose, reaching high impact energy, water lubrication is recommended.

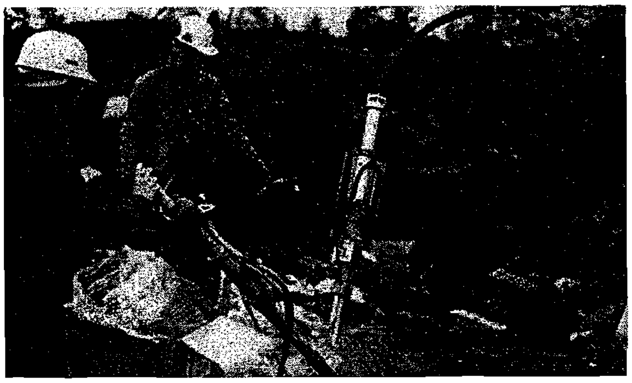


Photo 15.1. Robot charger.

At present, Nitro-Nobel A.B is developing new equipment for charging upholes with diameters of up to 165 mm. Of the two systems that are in experimental phase, *Charge-Pusher* and *Half-Pusher*, Fig. 15.2 shows the working principle of the latter.

Without going into detail, this device has a climbing mechanism with which, by upward movements, it pushes the charge ahead to the desired position. In each pushing movement an expansion element presses against the walls of the blasthole, retaining the climber in place while the piston rod forces the cartridge upwards which is held in place by a spider-like piece.

15.2.2 *ANFO type explosives*

Charging systems

Depending upon the capacities of the containers, the charging systems are classified as follows:

- Pneumatic chargers
- Charging trucks (Mix-Load and Mix-Pump)

The first system is mainly used in underground operations and small surface mines, whereas the second is exclusively for large mines and surface operations.

Pneumatic chargers. In these chargers, Fig. 15.3, the explosive is propelled through an antistatic, semiconductive hose by air pressure contained in a metal vessel or *pot* that is hermetically closed. The design

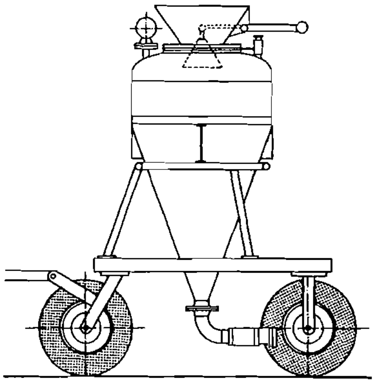


Fig. 15.3. Pneumatic charger.

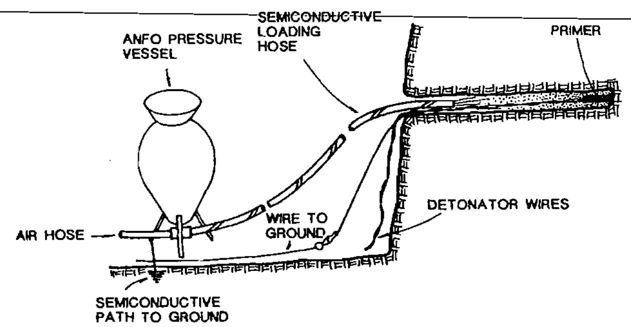


Fig. 15.4. Control of static energy in pneumatic loading.

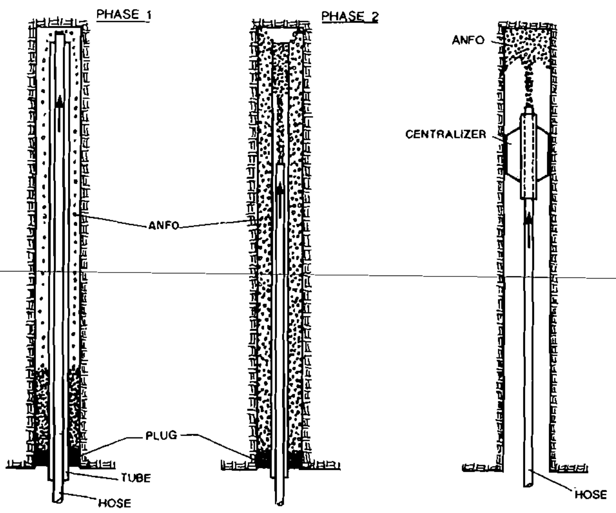


Fig. 15.5. Pneumatic loading of ANFO in upholes.

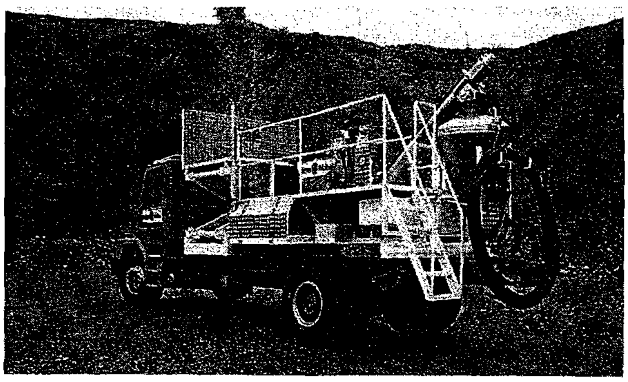


Photo 15.2. ANFO loader on vehicle.

consists in a funnel-shaped bottom, a cylinder-shaped body and another cone-like shaped of stainless steel that is corrosion resistant.

The capacity of these chargers varies from 100 to 750 liters, and when transported they are mounted individually on wheels or upon a vehicle, Photo 15.2. For the latter, the air is pressurized by compressor activated by the motor of the vehicle, which also has recipients of the explosive for the automatic recharging of the vessels, or a prepared space for ANFO sack storage when the refilling is done by hand.

When upholes are loaded in underground operations, the pressure of the vessel must be combined with the Venturi effect created by blowing pressurized air through the injector, to produce compressing of the ANFO prills in the bottom of the blasthole so that they will stick and not fall out.

The working pressures go from 0.15 to 0.3 MPa in the vessels, and from 0.2 to 0.35 MPa in the injectors.

This type of charging equipment is recommended for blastholes with diameters between 26 and 150 mm, unless they are upholes, where the diameters are limited to 100 mm.

The yield of the chargers depends upon the interior diameters of the hoses and their length, which should never be over 50 m, and the inclination of the boreholes. The maximum charging capacity oscillates between 2 and 4 tons. Apart from the equipment already described, there are lighter models on the market which can be transported by the operator himself, with a capacity of from 25 to 40 kg of ANFO.

These are used in underground operations to charge blastholes of 28 to 65 mm in diameter and basically consist of small vessels of polyethylene plastic with straps for their transport. They work with air pressures that go from 0.4 to 0.8 MPa and the charging capacity reaches 7 kg/min.

A very important aspect, from a safety point of view, is the elimination of the large amount of static electricity that is produced. In order to do this, it is necessary to properly connect the loading hose, made of semiconductive material, and properly ground the whole equipment, Fig. 15.4.

In the particular case of large diameter upholes, the traditional method of pneumatic loading, consisting of a lower closing plug and a charging tube has been progressively substituted by the direct method represented in Fig. 15.5, where the pressure given to the ANFO, which varies between 0.14 and 0.2 MPa, is sufficient to make the ANFO prills stick to the bottom of the holes giving charge densities of 0.95 to 1 g/cm^3. It is of vital importance in this system to have a correct design of the centralizer in the charging tube.

If there is water present in the blastholes, the loading can be done after placing a plastic liner.

The primers that are connected to downlines or to the detonator are usually placed in the bottom of the blastholes by means of a retainer with help of the loading hose itself.

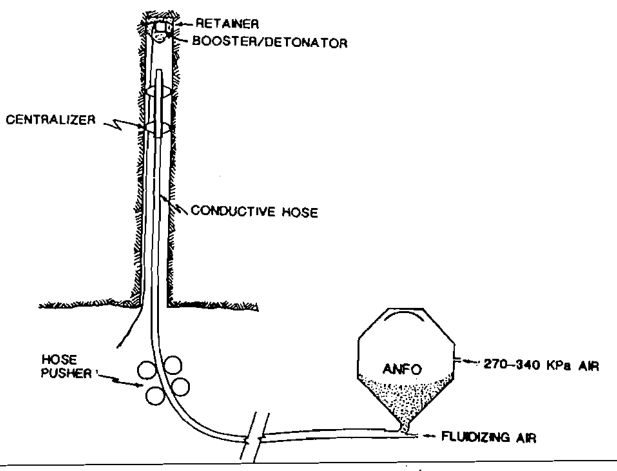


Fig. 15.6. Placing the primer in the bottom of a large diameter uphole before loading.

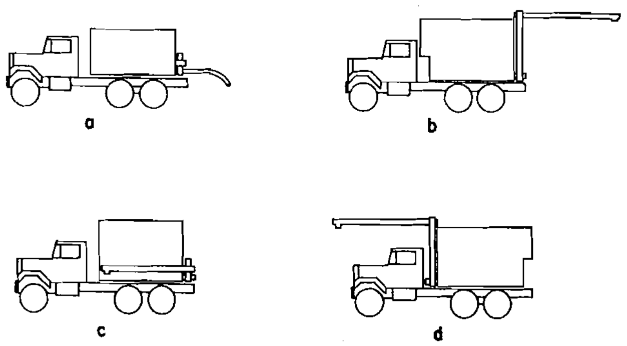


Fig. 15.7. Types of bulk loading trucks, (a) pneumatic delivery, (b, c, and d) auger delivery.

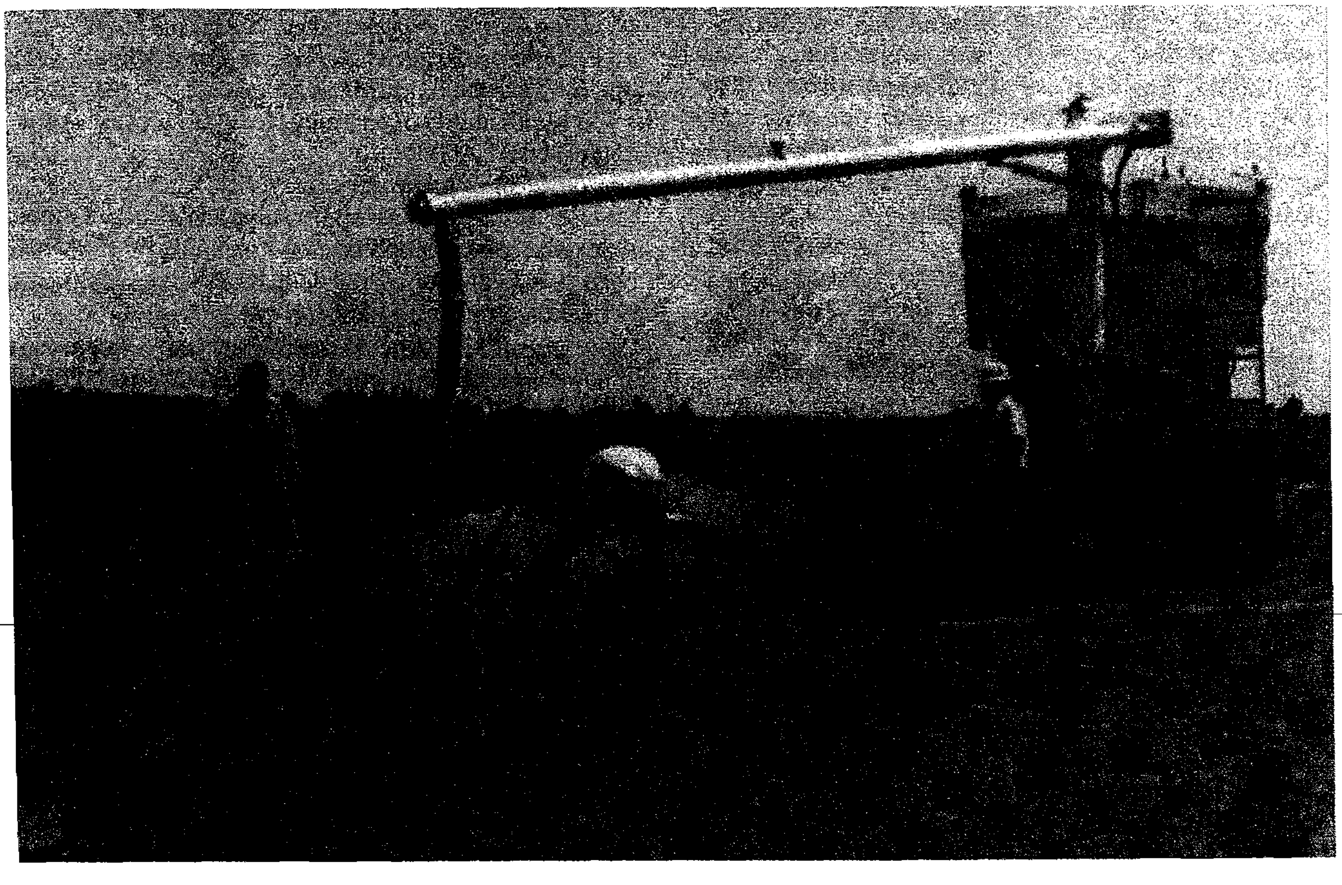


Photo 15.3. Bulk loading truck with helicodial auger (Courtesy of Amerind-MacKissic, Inc).

Bulk loading trucks. The types of tank trucks used for charging granular ANFO-type explosives are:

- Pneumatic delivery system
- Auger delivery, Fig. 15.7.

The first type of truck is the most used in Spain at the moment, and it consists of a closed aluminum deposit (AN hopper) with top and bottom V-shaped charge openings to aid in the descent of the explosive towards the conveyer or feed auger which conveys the ANFO for mixing and should be protected by an inverted V-trough which keeps the conveyer from holding the whole weight of the charge.

On the outside part of the deposit is a mechanism which regulates the height of the explosive on the feed auger, as well as a tachometer for the roller motor permittimg variations in the speed, dosifying the supply of the rotary air-lock feeder which discharges the explosive by air pressure through an antistatic hose to the inside of the blasthole.

The rotary air-lock feeder is composed of a drum wheel with plastic blades which also keeps the pressured air out of the ANFO bin.

The engine of the vehicle is connected to the hydraulic pumps that activate the feed auger and the rotary air-lock feeder, as well as the air compressor.

The loading hose is located in the back of the truck and is about 10 m in length which permits the charging of 3 or 4 blastholes from the same position when the truck is driven between two rows.

The problems with this system are the segregation of the aluminum when ALANFO is used, and the impossibility of loading Heavy ANFO.

The second model of truck has, at the bottom of the deposit and lengthwise, a helicoid auger that is also protected by deflecting plates.

This auger feeds another vertical one which then delivers the product to a third subhorizontal, pivoting boom auger. This last auger has a length of between 5 and 6 meters and covers a circular sector of 345°, loading blastholes through a flexible hose that are 6 or 7 m from the back part of the truck, Photo 15.3.

When the truck is between two rows of large-diameter blastholes, the number of these that can be loaded from one position is limited to one or two.

The loading flow of these trucks varies between 150 and 750 kg/min.

A more simple version of this truck is one called Side Auger Discharge System. In the back of the vehicle there is an inclined discharge auger that delivers the explosive to another swiveling boom auger of approximately 3 m in length. This covers a circular sector of 180° with manual rotation, as well as its elevation or lowering by means of a small hand winch. During transit the auger rests in a cradle along the lower left side of the body.

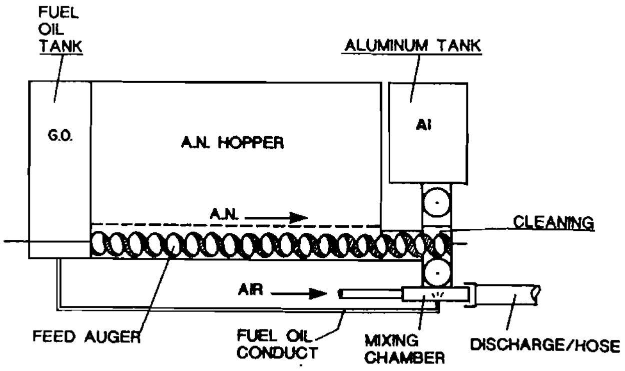


Fig. 15.8. Deposits on a mix-load truck.

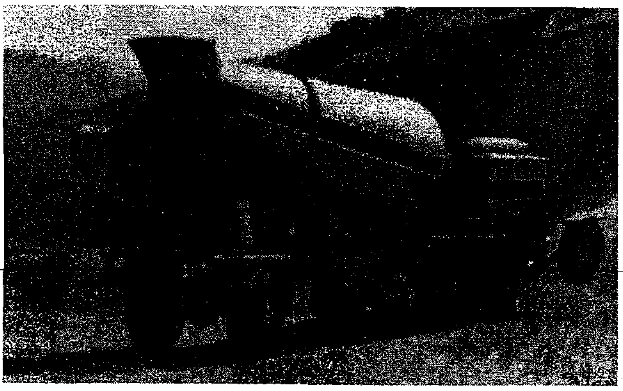


Photo 15.4. Bowl-type mix-load truck.

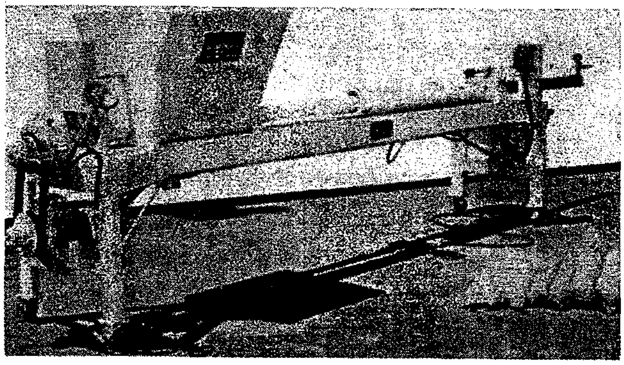


Photo 15.5. ANFO cartridges (Amerind Mackissic, Inc.).

During the last few years, there has been a progressive tendency towards trucks having an auger delivery system, owing to the following advantages:

- The possibility of charging Heavy ANFO as well as ANFO and ALANFO.
- Greater discharge rates, and
- Lower loss of ammonium nitrate and distillate vapor around the collars of blastholes.

Mix and load systems

Conventional Mix-Load trucks. These have a hopper of ammonium nitrate and a tank of fuel oil. If ALANFO or Heavy ANFO is required, there is also a third tank with the emulsion blasting agent or aluminum powder, Fig. 15.8.

Moments before loading the blastholes, the two or three components are mixed in the truck, in the desired proportions, and the resulting explosive is then delivered by either of the two systems described previously.

The hopper of ammonium nitrate is similar to those already mentioned. In the pneumatic discharge units the fuel is added with the air whereas in those of auger delivery, the fuel oil and other additives are delivered through the vertical auger.

Bowl-type Mix-Load trucks. These trucks are similar to concrete trucks with slight modifications to make them safe for mixing and charging bulk blasting agents. The components are placed in the bowl in adequate proportions and are mixed accordingly before being discharged.

The explosive obtained with these units is characterized by:

- Smaller errors in the overall chemical composition
- More uniform blending and, therefore,
- The energy outputs closely resembles those achieved in laboratories.

When compared with conventional mix-load trucks, bowl-type trucks offer the following advantages:

- Lower capital cost (about 30%).
- Higher discharge rates, close to 2.000 kg/min. (this is 2.5 to 4 times those obtained by conventional trucks).

On the other hand, bowl-type trucks have the following disadvantages:

- The truck must be positioned very close to the blasthole for loading, losing time in changing positions.
- Only one type of explosive can be charged each time, eliminating the possibility of selective charging.
- The quantity of explosive mixed must be exactly the amount required in order to avoid excess, which must be removed.
- The capacity of these trucks (approximately 11.5 t) is 25% less than conventional trucks.

Cartridged ANFO

When drilling 76 to 190 mm diameter blastholes and water appears, the explosive columns can be prepared with ANFO cartridges once the hole has been dewatered.

The packaging of ANFO is done with simple equipment consisting in a hopper, a one meter long tube, a feed auger and a piston system that works with pressurized air to achieve the required charge density that can reach 1.1 g/cm^3. The yield is around 3 cartridges per minute.

15.2.3 *Slurry and emulsion-type explosives*

Pump trucks for slurries and emulsions. These trucks are used for pumping explosives such as slurries and emulsions, and mixtures of emulsions with ANFO, whenever the solid phase of these mixtures is not more than 35%, because then the product would no longer be pumpable.

The physical consistency of these blasting agents is so high that for their pumping the injection of a liquid lubricant along the inside wall of the loading hose is usually necessary to reduce friction and facilitate easy, rapid pumping. It is important to use the lowest feasible amount of lubricant, and that it contribute to enhancing the effective explosion energy whenever possible.

Mix-pump trucks. A mix-pump truck is a mobile plant in which the products are continuously mixed and are pumped directly into the blastholes through a flexible hose. This system is quite versatil, as it allows variation in the compositions before charging begins. The vehicles have a capacity of between 5 and 15 t and are designed to produce at least two types of explosives, one for bottom charging and one for the column charge.

These mobile plants are very safe as the ingredients they carry are not explosive alone and they are mixed only instants before charging. On the other hand, quality control is more difficult than with pump trucks.

a) *Slurry mix-pump truck*

These trucks transport the following ingredients:

- A hot solution of ammonium nitrate, with or without oxidizers such as sodium nitrate, calcium nitrate, etc., thickened by gums. This solution is prepared at a static plant near the minesite.
- Ammonium nitrate in pourous prill form (optional).
- Liquid fuel-oil or a mixture of solid fuels that are called pre-mixes, with a percentage of aluminum as high as the required weight strength of the watergel.
- A cross-linking solution and a gassing agent.

The ingredients are put into the truck's mixing funnel from which they pumped into the blasthole through a flexible hose.

The charging rates vary between 80 and 350 kg/min.

Thickening and cross-linking starts as soon as the products are mixed so that the watergel is highly viscous by the time it enters the blasthole.

The gelling can be controlled by adjusting the cross-linking solution.

When the gelling ocurrs too rapidly, pumping difficulties appear, whereas if the gelling time is too long the slurry can become diluted or even dissolved before its viscosity permits it to resist the effect of the water present in the blastholes. The loading hose operator should be certain that there is a minimum agitation of the explosive

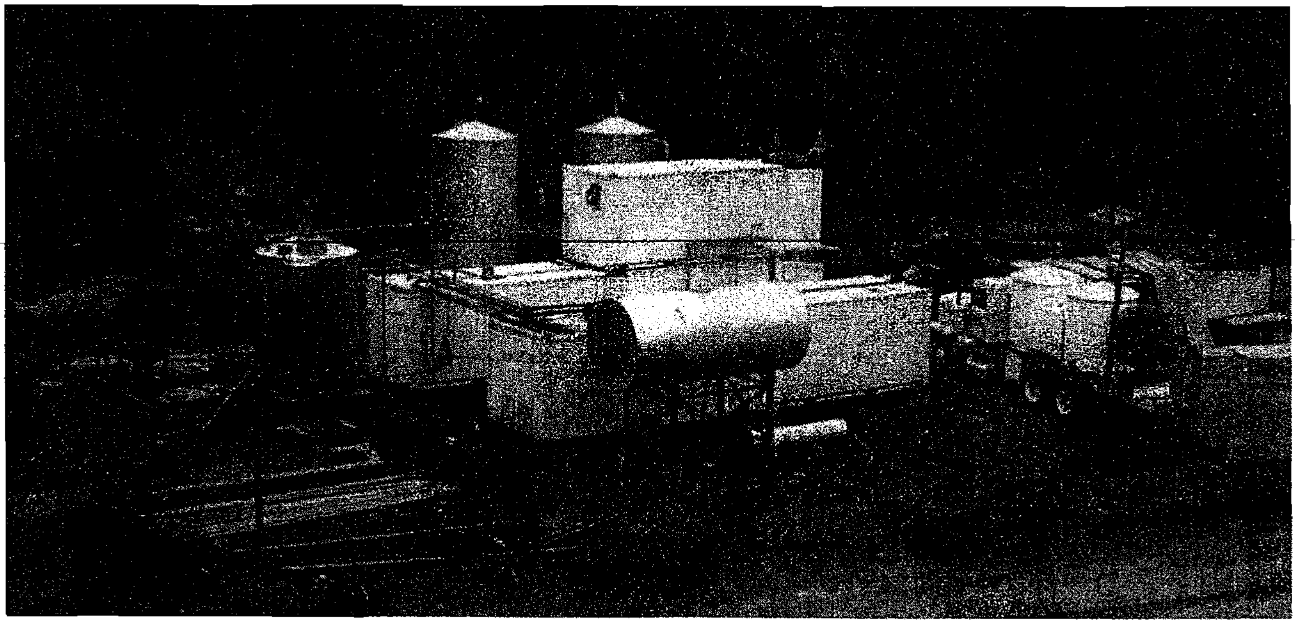


Photo 15.6. Static plant and pump truck (Nitro Nobel).

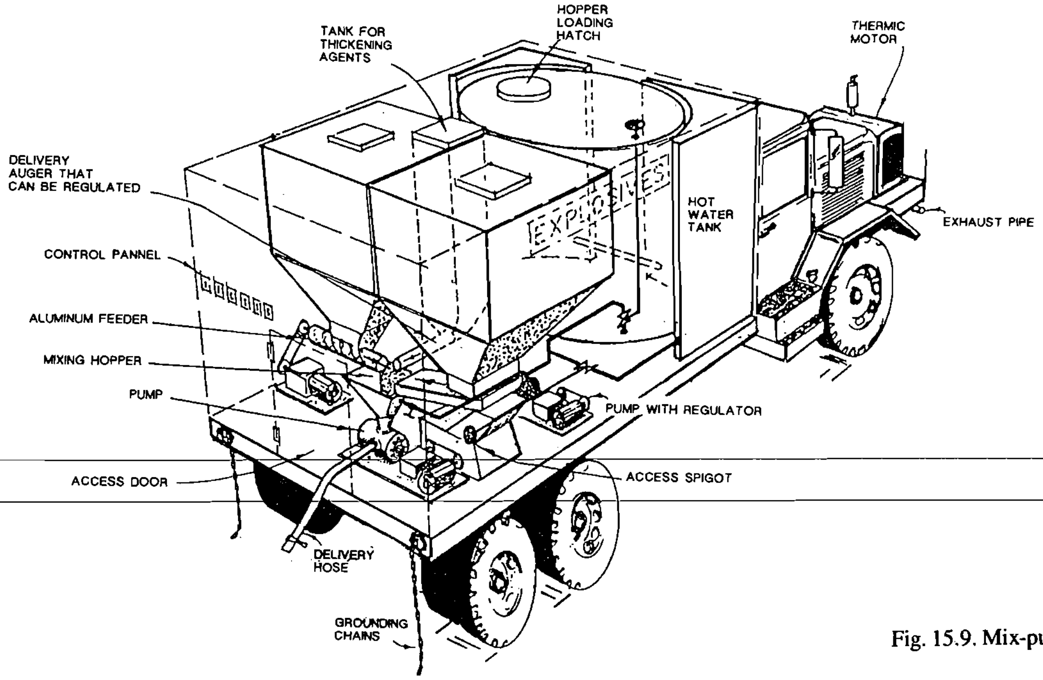


Fig. 15.9. Mix-pump truck (Ireco Inc.).

when it enters into contact with the water.

The proportion of gassing agent should be adjusted to give the slurry the required sensitivity and bulk strength. If the gassing is insufficient, a density in the botton of the column will be produced, reducing the optimum yield of the explosive. On the other hand, excessive gassing can reduce the density of the explosive making it float in the water. The flow of gassing solution can be controlled and can give slurries with a wide range of densities. This possibility is the basis of the technique called *Powerdecking*.

b) *Mix-pump trucks for charging emulsions and mixtures of emulsion/dry phase*

In this type of trucks, a continuous mixture of a saturated solution of oxidizers is produced, with an oil phase and some other ingredients in small amounts. The resulting product is pumped into the blasthole.

If a dry phase such as ANFO or ammonium nitrate prills are added to the mixture, it is important to ensure that the emulsion produced does not lose its pumpable qualities.

Pump trucks. When pump trucks are used, the blasting agent is previously manufactured in a static plant near or on the minesite.

The advantages of this system are:

– The static plant can be located in the center of the various points of consumption, supplying the slurry or the emulsion in several trucks, and

– The product is of higher quality than that produced in the mix truck.

Underground charging of slurried and emulsions

Loading blastholes in underground operations has different methods, depending upon the type of work at hand:

Drift driving. The motor-pump system used is customarily mounted on a small size vehicle that sometimes has a hydraulically powered man basket enabling the blaster to have access to the back holes, operating the pump with remote control.

The most popular types of pumps are those of diaphragm and those with auger which aspirates the explosive from the tanks which have a capacity of up to 500 kg and load it with a pressure of about 0.5 MPa, Photo 15.7.

The loading hoses are semi-conductive to eliminate static electricity and are introduced into the blastholes up to about 20 cm from the bottom, then pumping the explosive which gradually pushes the hose out of the hole until the desired charge height is reached. Initiation is usually achieved with a primer cartridge and an electric blasting cap, previously placed in the bottom of the hole.

The flow rates are comparable to those obtained with ANFO pneumatic chargers. Depending upon the pump speed, a 3 meter long blasthole with 41 mm in diameter can be charged in 6 to 10 seconds.

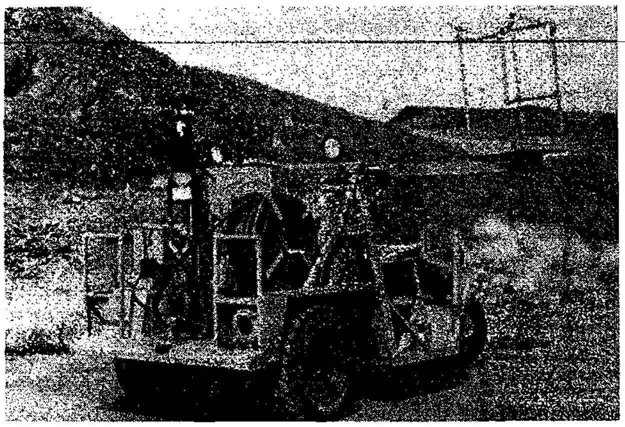


Photo 15.7. Charging equipment for development headings.

Photo 15.8. Pneumatic pump (Bill Lane Inc.).

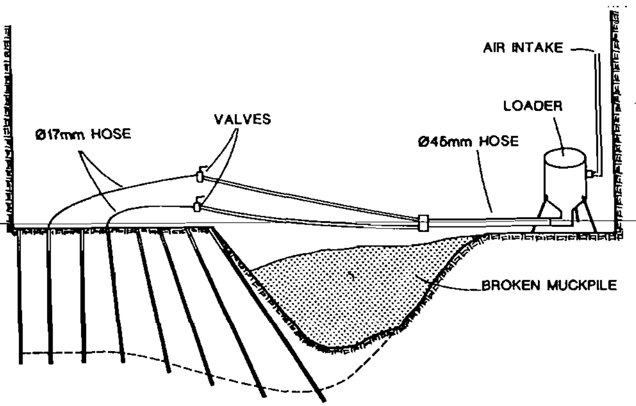


Fig. 15.10. Pumping equipment for loading explosives in shaft sinking.

Shaft sinking. Pressurized vessels are used, similar to those used with bulk ANFO. The discharge of the explosive through a main hose of 45 mm, reaches a flow rate of 77 kg/min, that is at the same time divided into 5 flexible hoses of 17 mm diameter which permits the loading of blastholes in a very short time, Fig. 15.10.

Production blasts. In production blasts with large diameter blastholes, more than 125 mm, there are two different charging situations: upholes and downholes.

a) *Downholes*. They are used in the operational methods of inverted craters and in levelling with long blastholes. Charging is carried out very easily because the explosive is pumped and descends by gravity to the bottom of the hole. The pump units have a modular design and a low profile to facilitate the transport of the explosive from the surface as well as in the mines. The exchangeable tanks of explosive are made of stainless steel with capacities of close to 2.000 kg. The pump, hose and the inclination hinge of the tank are hydraulically powered.

b) *Upholes*. The charging of upholes with blasting agents such a slurries and emulsions is even more difficult than with ANFO, as it is first necessary to apply a borehole plug to keep the explosive from falling out and, secondly, the product must have an adequate consistency for pumping. The latter seems to have been solved for emulsions by cooling.

As to plugs, there are various systems used. The first ones used a wooden plug with an interior tube that had a check valve with a brass anti-return ball, Fig. 15.11.

Plastic tubes have also been used to make up the explosive columns, and wooden plugs with holes that allowed the charge tubes to pass, Fig. 15.12.

More recently, a polyurethane foam uphole plug has been used, developed by C-I-L, Fig. 15.13.

And lastly, with repumpable emulsions, in which their viscosities can be adjusted to the work conditions by selection of surfactants, type of fuel and particle sizing,

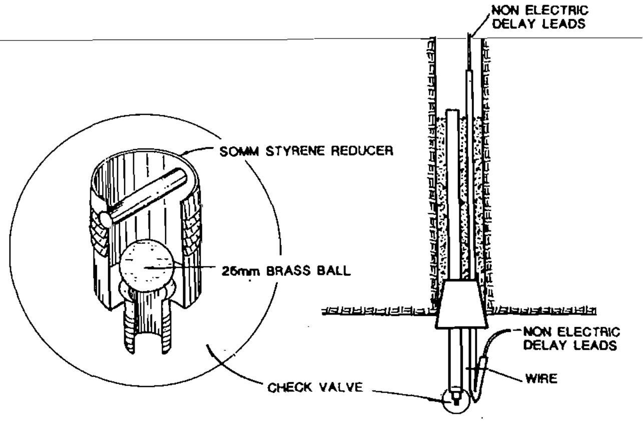


Fig. 15.11. Wooden plug with anti-return check valve.

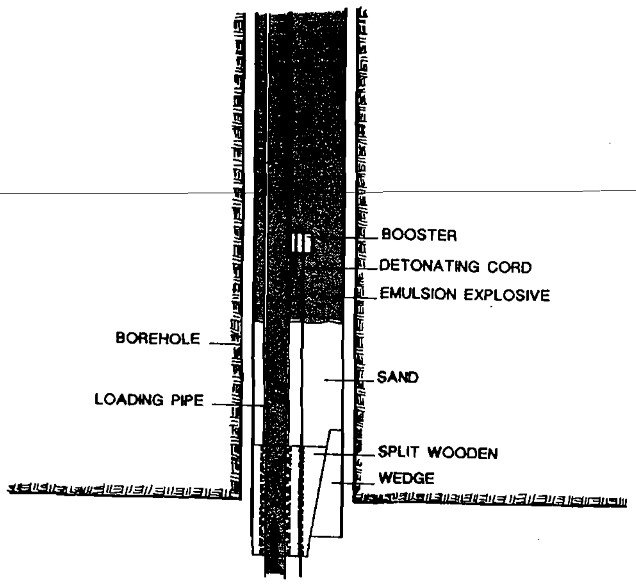


Fig. 15.12. Tubed charging with wooden plug.

inflatable lances have been tried with success. These devices have two inches of flexible hose with a rigid tube on one end, upon which an inflatable rubber bladder is mounted and inflated by pressurized air, Fig. 15.14.

The advantages of this system are its simplicity and low cost. It is quick and efficient, having been successfully tried in blastholes of up to 115 mm in diameter.

15.3 BLASTHOLE DEWATERING SYSTEMS

The dewatering of blastholes widens the field of application of bulk ANFO to those cases where the rock masses are saturated or currents, owing to rain, fill the blastholes with water; and also, when charging, the volume of hole drilled is used to better advantage.

The most used systems are classified as:

- Air operated pumps and,
- Submergible impellent pumps.

The first are applied to small and medium diameter (63 to 172 mm) blastholes with a maximum bench height of about 15 meters. Pressurized air supplied by compressors of the drilling rigs is used, which is introduced into the blastholes through a flexible plastic hose.

In some equipment, Fig. 15.15, the pushing effect is achieved when the obturator or plastic closing sleeve expands when the air pass through.

The pumping rates are approximately 50 to 80l/min.

The second dewatering system has a submergible impellent pump and a reel for the hose. The unit can be installed on a jeep-type vehicle or on the back of an ANFO charge truck. The reel and pump are hydraulically driven and the hydraulic fluid tubes of the latter are joined inside the water hose, enabling the whole ensemble to be lowered into the blasthole at a speed of approximately 1 m/s.

To avoid stoppage produced by coarse waste material, the pump should by placed at a few centimeters from the bottom.

Once the dewatering of the blastholes is finished, the mechanism of the drum wheel reverses to clean it of sand and waste that might have entered.

These units can dewater blastholes in a few seconds,

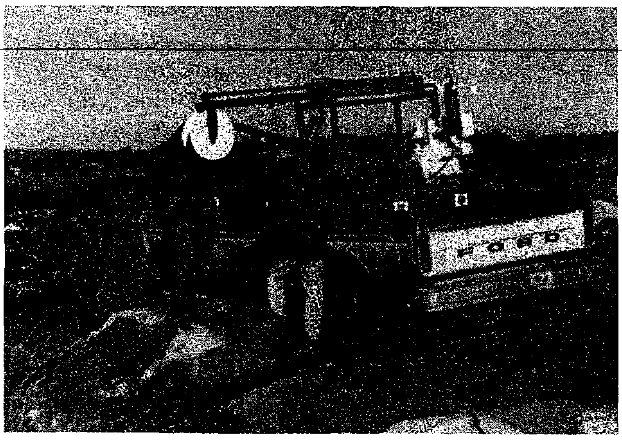


Photo. 15.9. Hydraulic dewatering pump (Swanson Eng. Inc.)

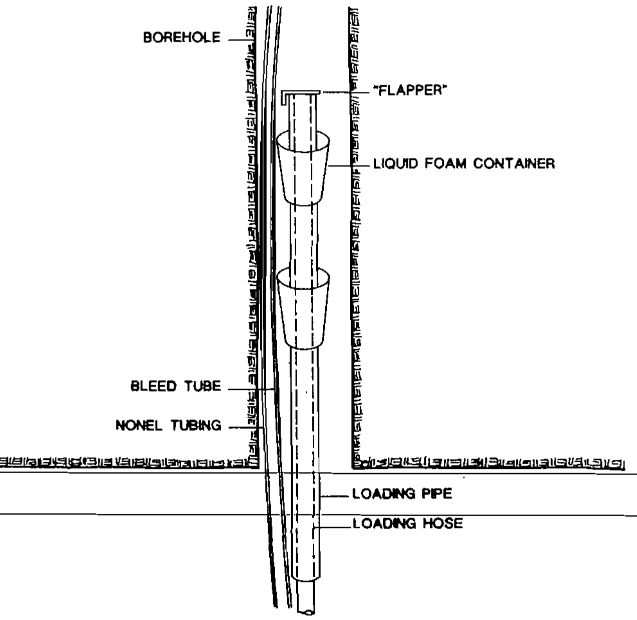


Fig. 15.13. Polyurethane foam uphole plug.

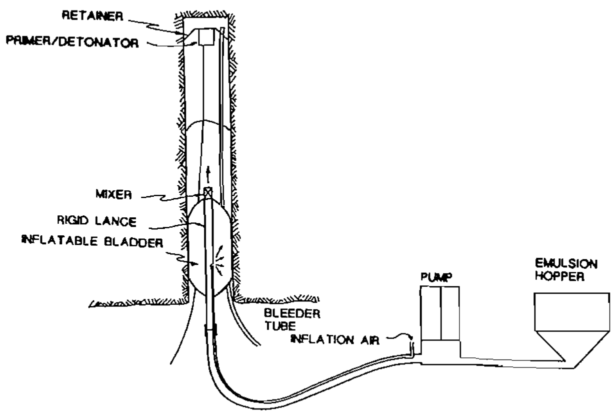


Fig. 15.14. Charging of a repumpable emulsion in an uphole with an inflatable lance.

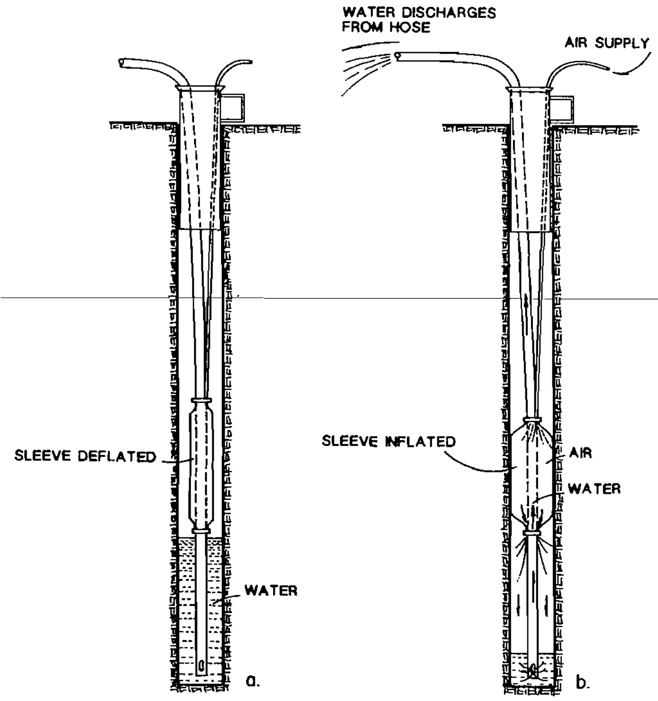


Fig. 15.15. Pneumatic pump.

Table 15.2.

Total elevation height (m)	Flow (l/min) Diameter 80 mm, Sist. Hid = 17 lpm, 11 MPa	Diameter 136 mm, Sist. Hid. = 23 lpm, 13 MPa	Diameter 187 mm, Sist. Hid. = 38 lpm, 13 MPa
7	246	472	1134
15	189	340	945
22	151	227	756
30	113	151	567
37	76	113	378

Table 15.3.

Blasthole diameters (mm)	Nominal diameter of plastic liner (mm)
152	166
200	216
229	248
251	274
270	293
279	299
311	337
381	407

owing to the strong pumping rates, Table 15.2, permitting use of the plastic sleeves and charging before the water enters again.

The type of plastic used should be flexible and resistant so that it will not tear when in contact with the rock, recommending that it be of 600 to 1.000 gage, depending upon each case.

The liners or plastic sleeves, which contain the bulk explosive, should have a diameter that is slightly more than that of the holes, Table 15.3, so that the volume of rock drilled can be used to maximum advantage and achieve a good adaptation of the charge.

Photo 15.10. Preparation of the primer charge in a plastic sleeve.

REFERENCES

Amerind-Mackissic, Inc.: *Technical Information.* 1986

Bauer, A.: *Trends in Explosives, Drilling and Blasting.* CIM Bulletin, February, 1974.

Bauer, A. et al.: *Drilling and Blasting in Open Pits and Quarries.* 1980.

Bill Lane, INC.: *Lane Pump.* 1986.

Champion, M.M.: *Explosives Loading Equipment.* Underground Mining Methods Handbook. AIME, 1982.

Dannenberg, J.: *Contemporary history of industrial explosives in America.*

Day, P.R. & D.K. Joyce: *Loading explosives in large diameter upholes.* SEE, 1988.

Giorgio, C.: *Evolución de los Explosivos en los Treinta Ultimos Años.* Rocas y Minerales.

Gustafsson, R.: *Swedish Blasting Technique.* SPI, 1973.

Hagan, T.N.: *Charging and Dewatering Equipment.* AMF. 1985.

Irvine, J.C.: *Pillar recovery at the Pea Ridge Mine.* Mining Engineering. September, 1976.

Jerberyd, L.: *Half-pusher – A method to charge large diameter upholes.* Swedish Mining Research Foundation, 1985.

Legorburu, V.: *Sistemas Mecanizadas de Carga de Explosivos en Proyectos Subterráneos.* I Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, 1986.

Lopez Jimeno, C.: *Desagüe y Drenaje de Explotaciones a Cielo Abierto.* IV Curso sobre Mantenimiento y Servicios en Minería a Cielo Abierto. Fundación Gómez-Pardo. 1984.

Mairs, D.B. & R.B. Tupling: *Application of emulsion explosives in underground.* CIM Meeting, 1985.

Michaud, P. & A. Laveault: *Essai d'un systeme de chargement en vrac pour emulsions aux Mines d'Amiente Bell.* SEEQ, 1984.

Nitro-Nobel: *ANFO Mixing and Charging Equipment.* 1986. *Swedish Methods for Mechanized Blasthole Charging.*

Puntous, R.: *Mèthodes Modernes de Chargement des Explosifs.* Industrie Minèrale – Les Techniques. Fèvrier, 1984.

Sharpe, K.R.: *Plugging and loading upholes at La Mine Bosquet.* CIL Inc. 1986.

Swanson Engineering Inc.: *Blasthole dewatering – Cuts costs.*

Union Española De Explosivos: *Tendencias Actuales en el Almacenamiento, Transporte y Carga Mecanizada de Explosivos en la Minería a Cielo Abierto.* Jornadas Técnicas, UEE.

VME-Nitro Consult, Inc.: *Pneumatic Cartridge Charging.*

Yetter, A. & R. Malo: *The evolution of loading 4.5 inch diameter upholes at Kidd Creek No. 1 Mine.* SEE. 1984.

CHAPTER 16

Mechanisms of rock breakage

16.1 INTRODUCTION

During the detonation of an explosive charge inside rock, the conditions presented are characterized by two phases of action:

1st. phase. A strong impact is produced by the shock wave linked to the Strain Energy, during a short period of time.

2nd. phase. The gases produced behind the detonation front come into action, at high temperature and pressure, carrying the Thermodynamic or Bubble Energy.

Since the decade of the fifties, many theories have been developed to explain the behavior of rocks under the effect of an explosion; even nowadays it still remains a problem to be solved and defined in the technology of application of explosives to breakage. Without entering into detail, the different mechanisms of rock breakage that have been identified in blasting up to now are exposed in the following paragraphs.

16.2 ROCK BREAKAGE MECHANISMS

In the fragmentation of rocks with explosives at least eight breakage mechanisms are involved, with more or less responsability, but they all exert influence upon the results of the blastings.

16.2.1 *Crushing of rock*

In the first instants of detonation, the pressure in front of the strain wave, which expands in cylindrical form, reaches values that well exceed the dynamic compressive strength of the rock, provoking the destruction of its intercrystalline and intergranular structure.

The thickness of the so called *crushed zone* increases with detonation pressure of the explosive and with the coupling between the charge and the blasthole wall. According to Duvall and Atchison (1957), with high strength explosives in porous rocks it might reach a radius of up to 8 D, but it is normally between 2 and 4 D.

In Fig. 16.1. the variations in compressive stresses generated by two fully-coupled charges are shown. The crushing of the rock is produced at a pressure of 4 GPa, so the curve of the explosive (A) which produces a tension of 7 GPa on the blasthole wall has a very sharp decrease in peak stress due to the large increase in surface area during the pulverization of the rock. As explosive (B) does not increase the surface area by crushing, it has a slower rate of stress decay than (A).

According to Hagan (1977), this breakage mechanism consumes almost 30% of the energy transported by the strain wave, only contributing a very small volume to the actual rock fragmentation, around 0.1% of the total volume corresponding to the normal breakage per blasthole. Therefore, there is no incentive to use high explosives that generate high stresses on the blasthole walls: which would even make it advisable to decouple the charges and increase *EB* in detriment of *ET*.

16.2.2 *Radial fracturing*

During propagation of the strain wave, the rock surrounding the blasthole is subjected to an intense radial compression which induces tensile components in the tangential planes of the wave front. When the tangential strains exceed the dynamic tensile strength of the rock, the formation of a dense area of radial cracks around the crushed zone that surrounds the blasthole is initiated, Fig. 16.2.

The number and length of these radial cracks increase with:

1. The intensity of the strain wave on the blasthole wall or on the exterior limit of the crushed zone, and

2. The decrease in dynamic tensile strength of the rock and the attenuation of the Strain Energy.

Beyond this inner zone of intense fracturing, some of the cracks extend noticeably and are symmetrically distributed around the blasthole. The propagation velocity of the cracks is from 0.15 to 0.40 times that of the strain wave, although the first microcracks are developed in a very short time, around 2 ms.

When the rock has natural fractures, the extension of the cracks is closely related to these. If the explosive columns are intersected lengthwise by a pre-existing crack, these will open with the effect of the strain wave and the development of radial cracks in other directions will be limited. The natural fractures that are parallel to the blastholes, but at some distance from them, will interrupt the propagation of the radial cracks, Fig. 16.3.

16.2.3 *Reflection breakage or spalling*

When the strain wave reaches a free surface two waves are generated, a tensile wave and a shear wave. This occurs when the radial cracks have not propagated farther

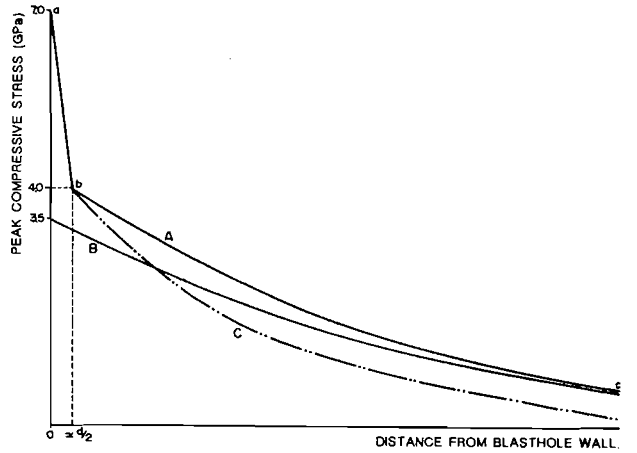


Fig. 16.1. Variation of peak compressive stress with distance from blasthole wall (Hagan).

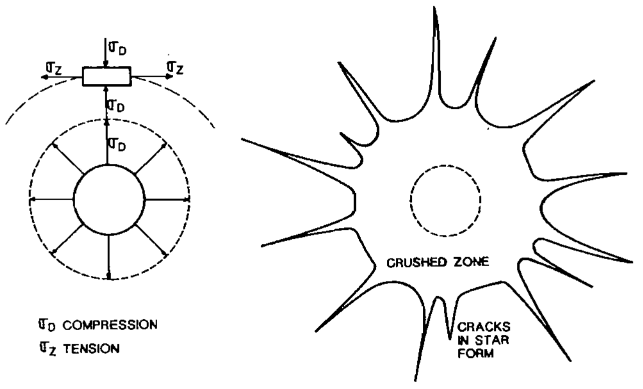


Fig. 16.2. Radial fracturing.

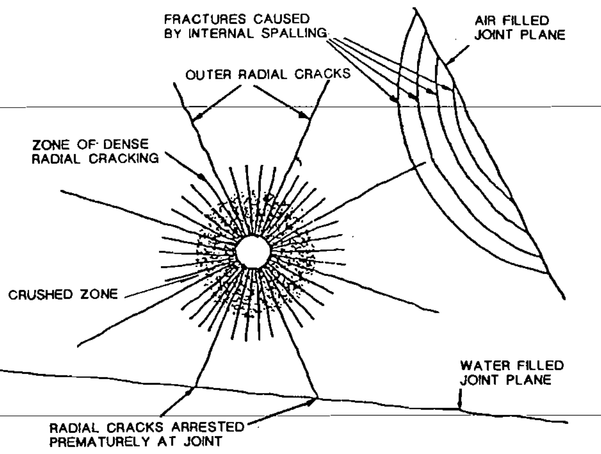

FRACTURES CAUSED BY INTERNAL SPALLING
AIR FILLED JOINT PLANE
OUTER RADIAL CRACKS
ZONE OF DENSE RADIAL CRACKING
CRUSHED ZONE
WATER FILLED JOINT PLANE
RADIAL CRACKS ARRESTED PREMATURELY AT JOINT

Fig. 16.3. Radial fracturing and breakage through reflection of the strain wave.

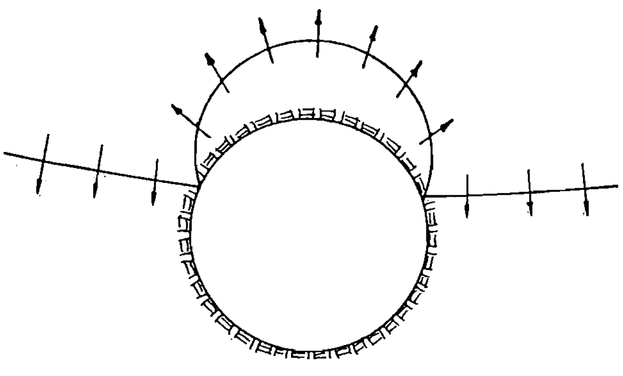


Fig. 16.4. Reflection of a wave upon a cylindrical cavity.

than one third the distance between the charge and the free face. Although the relative magnitud of the energies associated with the two waves depends upon the incident angle of the compressive strain wave, the fracturing is usually caused by the reflected tensile wave. If the tensile wave is strong enough to exceed the dynamic strength of the rock, the phenomenon known as spalling will come about, back towards the interior of the rock. The tensile strengths of the rock reach values that are between 5 and 15% of the compressive strengths.

The front of the reflected wave is more convex than that of the incident wave, which means that the dispersion index of the tensile wave energy is much larger when the surface is cylindrical, such as that of the central blasthole of a cut instead of when there is a plane as in bench blasting, Fig. 16.4.

This mechanism does not contribute much to the global fragmentation process, estimating that eight times more explosive charge would be necessary if rock were to be fragmented solely by reflected waves. However, in the inner discontinuities of the rock mass which are close to the charge, less than 15 D, and are not infilled with meteorized material, the effect of the reflected waves is more important due to the difference in impedances.

When excavating inclined ramps or shafts by blasting, it must be checked that the empty blastholes are not be filled with water in order to achieve the benefits of this mechanism of breakage.

16.2.4 *Gas extension fractures*

After the strain wave passes, the pressure of the gases cause a quasi-static stress field around the blasthole. During or after the formation of radial cracks by the tangential tensile component of the wave, the gases start to expand and penetrate into the fractures. The radial cracks are prolonged under the influence of the stress concentrations at their tips. The number and length of the opened and developed cracks strongly depend upon the pressure of the gases, and a premature escape of these due to insufficient stemming or by the presence of a plane of weakness in the free face could lead to a lower performance of the explosive energy.

16.2.5 *Fracturing by release-of-load*

Before the strain wave reaches the free face, the total energy transferred to the rock by initial compression varies between 60 and 70% of the blast energy (Cook et al. 1966). After the compressive wave has passed, a state of quasi-static equilibrium is produced, followed by a subsequent fall of pressure in the blasthole as the gases escape through the stemming, through the radial cracks

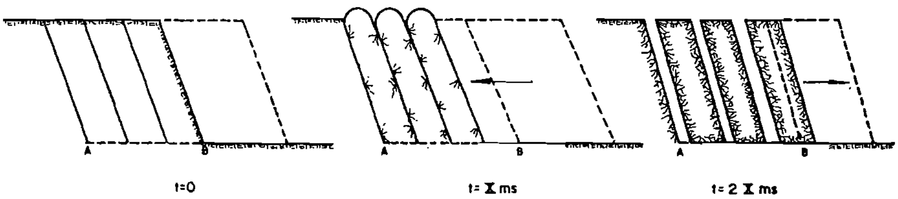


Fig. 16.5. Separation of layers of compressible medium by release-of-load.

and with rock displacement. The stored Stress Energy is rapidly released, generating an initiation of tensile and shear fractures in the rock mass. This affects a large volume of rock, not only in front of the blastholes but behind the line of the blast cut as well, having registered damages in up to dozens of meters away, Fig. 16.5.

16.2.6 *Fracturing along boundaries of modulus contrast of shear fracturing*

In sedimentary rock formations when the bedding planes, joints etc., have different elasticity modulus or geomechanic parameters, breakage is produced in the separation planes when the strain wave passes through because of the strain differential in these points, Fig. 16.6.

16.2.7 *Breakage by flexion*

During and after the mechanisms of radial fracturing and spalling, the pressure applied by the explosion gases upon the material in front of the explosive column makes the rock act like a beam embedded in the bottom of the blasthole and in the stemming area, producing the deformation and fracturing of the same buy the phenomena of flexion, Fig. 16.7.

16.2.8 *Fracture by in-flight collisions*

The rock fragments created by the previous mechanisms and accelerated by the gases are projected towards the free face, colliding with each other and thereby producing additional fragmentation which has been demonstrated by ultra-speed photographs (Hino, 1959; Petkof, 1969).

16.3 TRANSMISSION OF THE STRAIN WAVE THROUGH THE ROCK MASS

As shown beforehand, the Detonation Pressure can be expressed by the following simplified equation:

$$PD = \frac{\rho_e \times VD^2}{4}$$

where: PD = Detonation pressure (kPa), ρ_e = Explosive density (g/cm^3), VD = Detonation velocity (m/s).

The maximum Pressure Transmitted to the rock is the equivalent of:

$$PT_m = \frac{2}{1 + n_z} PD$$

where n_z is the relationship between the impedance of the explosive and that of the rock:

$$n_z = \frac{\rho_e \times VD}{\rho_r \times VC}$$

where: VC = Propagation velocity of the waves through rock mass (m/s), ρ_r = Rock density (g/cm^3).

This means that the explosive wave is better transmitted to the rock when the impedance of the explosive is close to that of the rock, given that n_z will tend towards 1, while PT will simultaneously tend towards PD. The pressure of the wave inside the rock decreases with the law of exponentials, so the radial stress generated at a determined distance will be:

$$\sigma_i = PB \times \left[\frac{r_b}{DS}\right]^x$$

where: σ = Radial compressive stress, PB = Pressure on the blasthole wall, r_b = Radius of the blasthole, DS = Distance from the center of the blasthole to the point in study, x = Exponent of the law of absorption which, for cylindrical charges is near 2.

If the wave encounters diverse material in its path, with different impedances and in correspondance with separating surfaces that can be in contact or separated by air or water, the transmission of the strain wave will be governed by the ratios of the acoustic impedances of the different types of rock, where part of the wave energy is transferred in the material and at the same time some is reflected back, as a function of the ratio.

When the impedances of the mediums are equal ($\rho_{r2} \times VC_2 = \rho_{r1} \times VC_1$), a large part of the energy will be transmitted and the rest will be reflected, arriving at the limit when ($\rho_{r2} \times VC_2 \ll \rho_{r1} \times VC_1$) as for example, between rock and air, where almost all of the energy will be reflected back as a tensile wave which could be especially important in the breakage of the rock.

The abovementioned is valid for the wave pressures as well as for the transmitted energies. If the ratio of characteristic impedances for the two materials is:

$$n'_z = \frac{\rho_{r1} \times VC_1}{\rho_{r2} \times VC_2}$$

the following will be obtained:

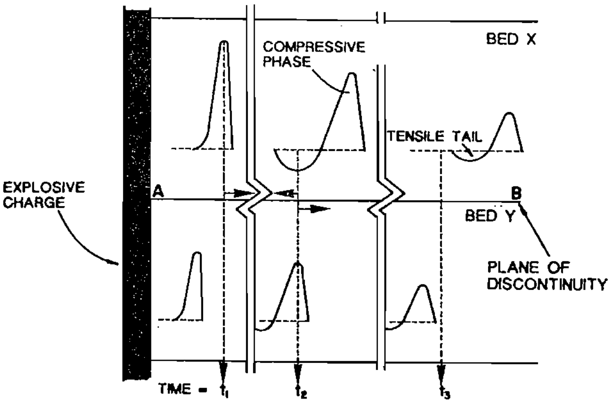


Fig. 16.6. Shear Fracturing (Hagan).

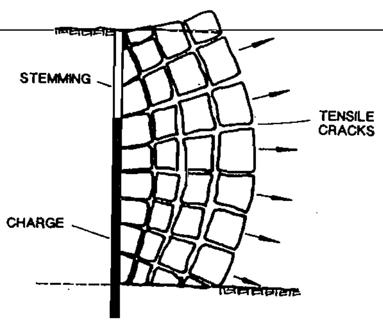


Fig. 16.7. Mechanism of breakage by flexion.

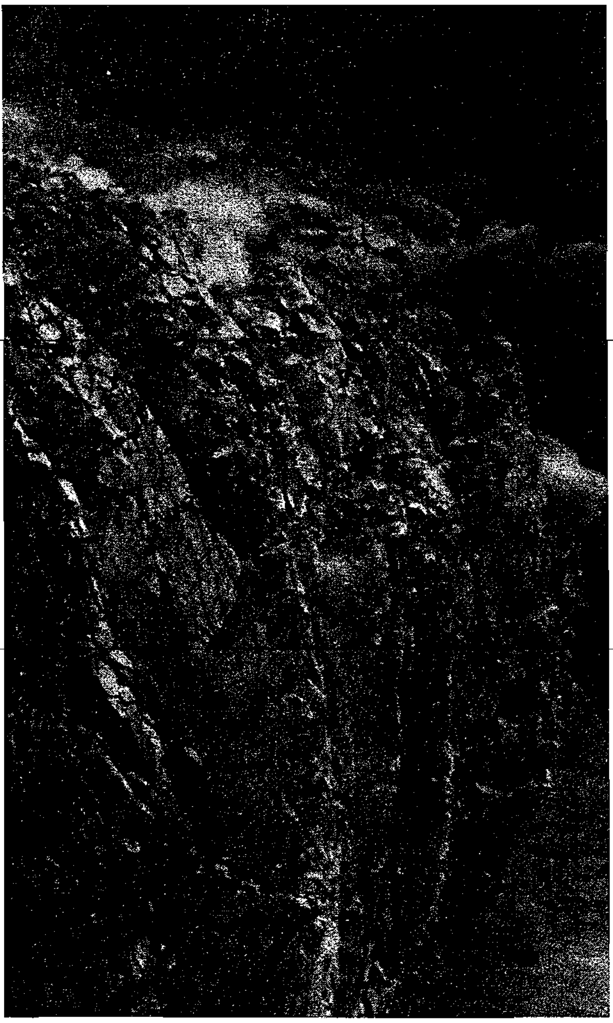


Photo 16.1. Rock breakage by flexion.

$$PT = 2\frac{PI}{(1 + n'_z)}$$

$$PR = PI\frac{(1 - n'_z)}{(1 + n'_z)}$$

where: PI = Pressure of the incident wave, PT = Pressure of the transmitted wave, PR = Pressure of the reflected wave.

16.4 ENERGETIC YIELD OF THE BLASTINGS

The performance of explosives upon the rocks is, therefore, a group of elemental actions which perform simultaneously in a few miliseconds, associated with the effects of the strain wave which transports the *Stress Energy*, and with the effects of the explosion gases or *Bubble Energy*, Fig. 16.8.

The total energy developed by the explosive and measured by the method proposed by Cole can thus be expressed as the sum of these two components.

$$ETD = ET + EB$$

where:

$$ET = \frac{K_1}{Q}\int P^2 \times \mathrm{dt}\ (\mathrm{cal/g})$$

$$EB = \frac{K_2}{Q} \times T_e^3\ (\mathrm{cal/g})$$

The estimates carried out by Hagan (1977) have demonstrated that only a 15% of the total energy generated in the blasting is used as a working tool in the mechanisms of rock fragmentation and displacement.

Rascheff and Goemans (1977) have established a model that theoretically distributes the energy, as represented in Fig. 16.9, from tests made upon cubic blocks of rock placed underwater in swimming pools. These investigators assure that approximately 53% of the explosive energy is associated with the strain wave. This value depends upon the conditions of the experiment and very different results can be found that go from 5 to 50% of the total energy, depending upon the various types of rock that are to be fragmented and the explosives used.

Therefore, in hard rock the Strain Energy of a breaking explosive is more important in fragmentation than the Bubble Energy, and the contrary is true for soft, porous or fissured rocks and in low density explosives.

From the tests carried out by Rascheff and Geomans, Table 16.1 summarizes the energy distribution of the strain wave.

It can be observed that in conventional bench blastings a large part of the strain wave energy is transformed into seismic energy which causes ground vibrations to which some of the gas energy must be added.

The data exposed are quite in accordance with that

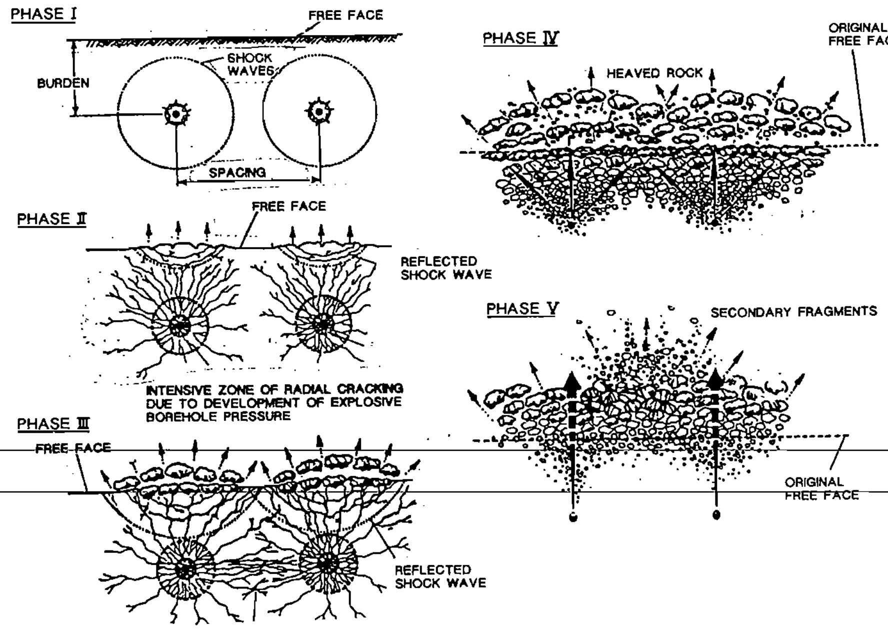
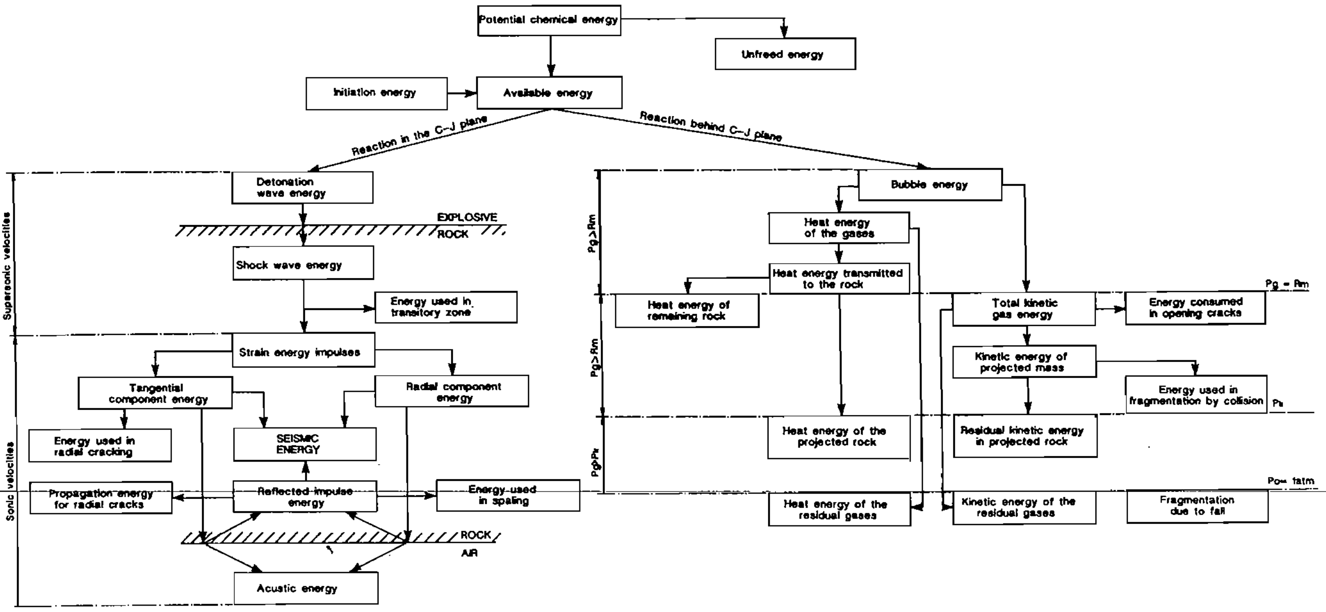


Fig. 16.8. Summary of the breakage mechanisms.


Potential chemical energy
Unfreed energy
Initiation energy
Available energy
Reaction in the C–J plane
Reaction behind C–J plane
Supersonic velocities
Sonic velocities
Detonation wave energy
EXPLOSIVE
ROCK
Shock wave energy
Energy used in transitory zone
Strain energy impulses
Tangential component energy
Radial component energy
Energy used in radial cracking
SEISMIC ENERGY
Propagation energy for radial cracks
Reflected impulse energy
Energy used in spalling
ROCK
AIR
Acustic energy
Pg>Rm
Pg>Rm
Pg>Pk
Bubble energy
Heat energy of the gases
Heat energy transmitted to the rock
Heat energy of remaining rock
Total kinetic gas energy
Energy consumed in opening cracks
Pg = Rm
Kinetic energy of projected mass
Energy used in fragmentation by collision
Pk
Heat energy of the projected rock
Residual kinetic energy in projected rock
Po= 1atm
Heat energy of the residual gases
Kinetic energy of the residual gases
Fragmentation due to fall

Pg – Maximum gas pressure on blasthole wall
Pk – Final pressure of expanding gases upon fragmented rock
Rm – Strength of original blasthole wall

Fig. 16.9. Distribution model of the explosive energy in a blast.

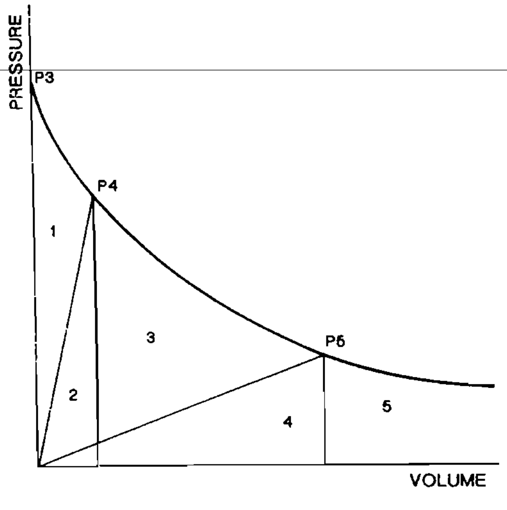


Fig. 16.10. Pressure/volume diagram of explosion product gases showing partition of energy in blasting.

Table 16.1. Distribution of shock wave energy.

	Granite block with infinite confinement	Conventional bench blasting of granite	Granite block submerged in water
Pulverization	15%	15%	15%
Primary radial cracking	3%	3%	2%
Crack extension	0%	16%	39%
Energy transmitted	82%	34%	22%
Useful energy	18%	34%	56%

Table 16.2

Zone	Energy
1	Kinetic component of shock energy
2	Strain component of shock energy
1 + 2	Brissance energy
3 + 4	Energy released during crack propagation
2 + 3	Fragmentation energy
4	Strain energy in burden at time gases escape
1 + 2 + 3 + 4	Blast energy
5	Heave energy
1 + 2 + 3 + 4 + 5	Total available energy or absolute strength value

obtained by other investigators such as Mancini and Occella.

It should not be forgotten that, in order to obtain optimum results in the blastings, it is not only necessary to fragment the rock but also to cause swelling and displace it a determined distance. For this reason, in the latter stages the gases also play a decisive role.

Lowends' used a simplified model of explosive/rock interaction to describe the partition of explosive energy in the process of rock blasting. The energy is partitioned into different zones that are related to the pressure/volume expansion of the gases during the different phases of blasting. An illustration of this partition of energy is given in Fig. 16.10.

The energies associated with the different zones given in the figure are, as follows:

When the explosive detonates in the blasthole, the high pressure gases at the initial or explosion state P3 send a shock wave into the rock. The strains from this shock near the blasthole are greater than the dynamic compressive and shear strength of the rock. They cause varying amounts of rock compression and crushing in the surrounding area of the blasthole depending upon the strength and stiffness of the rock. With rock compression and crushing the volume of the blasthole increases and the pressure decreases until the strain in the rock balances the pressure. This is shown as P4 on the pressure/volume curve of Figure 16.10, and is called blasthole equilibrium state. During the expansion, the work being done by the explosive is called brissance energy and consists of the strain energy stored in the rock (Zone 2) and the kinetic energy of the shock wave (Zone 1). The kinetic shock energy is essentially lost as useful work during the blasting process and appears as crushed rock surrounding the blasthole and as seismic waves propagated into the ground.

The strains in the rock coming from the residual blasthole pressure P4 cause fracture. The explosion product gases enter at least the cracks existing between the hole and the free face, resulting in fragmentation and possibly contributing to the heave. When the gases reach the free face through the burden, the process ends more or less abruptly. The pressure of the gases at escape is shown at P5 in Figure 16.10. During escape, the burden is compressed by the gas in the cracks with a strain energy stored in the rock (Zone 4). This energy has little influence on fragmentation and heave.

The energy from Zones 2 and 3 is the most useful in rock blasting and is called Fragmentation Energy.

At the time of escape, some of the energy in the gases (Zone 5) moves the burden and represents heave energy. The rest of this energy is lost as heat and noise in the escaping gases.

Although this model of energy partition oversimplifies the blasting process, it gives valuable insight into where the energy goes during the various phases of the process. It also provides approximate comparisons of the magnitude of the energy fractions used in the various phases of the blasting process as the explosive gases expand from the initial pressure in the blasthole to atmospheric pressure. Not all of the available energy is useful in fragmentation and heave. It may be possible to improve the efficiency of the blasting process by using explosives, whether ideal or not, that are designed to keep energy losses at a minimum.

REFERENCES

Ash, R.L.: *The Mechanics of Rock Breakage.* Pit and Quarry, no. 56, 1963.

Duvall, W.I. & T.C. Atchison: *Rock Breakage by Explosives.* U.S.B.M. RI 5356, 1957.

Hagan, T.N.: *Rock Breakage by Explosives.* Proc. National Symposium on Rock Fragmentation. Australian Geomechanics Society. Adelaide, Feb. 1973.

Hagan, T.N. & G.D. Just: *Rock Breakage by Explosives. Theory, Practice and Optimization.* Proc. Congress International Society of Rock Mechanics. Vol. II, 1974

Hagan, T.N.: *Rock Breakage by Explosives.* 6th Symposium on Gas Dynamics of Explosives and Reactive Systems. Stockhlom, 1977,

Land, L.C. & R.F. Favreau: *A modern approach to open pit blast design and analysis.* CIM Bulletin. June, 1972.

Lopez Jimeno, C.: *Los Mecanismos de Fragmentación con Explosivos y la Influencia de las Propiedades de las Rocas en los Resultados de las Voladuras.* I Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, 1986.

Rascheef, N. & P. Goemans: *Contribution à l'etude quantitative de l'energie consommée dans la fragmentation par explosif.* Oct.-Dec., 1977.

Thum, W.: *Quantite d'energie requise pour L'extraction et la fragmentation des roches au moyen d'explosives.* Explosifs, 1972.

CHAPTER 17

Rock and rock mass properties and their influence on the results of blasting

17.1 INTRODUCTION

The materials of which rock masses are made possess certain physical characteristics that are a function of their origin and of the posterior geological processes which have affected them. The whole of these phenomena make up a certain environment, a particular lithology with heterogeneities caused by the added polycrystalline minerals and by the discontinuities of the rock matrix (pores and fissures): and by a geological structure in a characteristic state of stress, with a large number of structural discontinuities such as bedding planes, fractures, diabases, joints, etc.

17.2 ROCK PROPERTIES

17.2.1 *Density*

The densities and strengths of rocks are normally quite well correlated. In general, low density rocks are deformed and broken quite easily, requiring relatively low energy factors, whereas dense rocks need a higher quantity of energy to achieve a satisfactory fragmentation, as well as a good displacement and swelling.

In high density rocks, the following measures should be taken to ensure adequate heavy energy:

- Increase the drilling diameter in order to elevate the blasthole pressure, $PB = k \times VD^2$, where VD is the detonating velocity of the explosive.
- Reduce the pattern and modify the initiation sequence.
- Improve the effectivity of the stemming to increase the time of gas performance and make certain that they escape through the free face instead of through the stemming.
- Use explosives with high bubble energy EB.

17.2.2 *The dynamic strengths of the rocks*

The static compressive RC and tensile RT strengths are initially used as indicative parameters of the suitability of the rock for blasting. The Index of Blastability was defined (Hino, 1959) as the relationship 'RC/RT', the larger the value, the easier the fragmentation.

The rational treatment of the existing problems require taking into consideration the dynamic strengths, as these increase with the index of the charge (Rinehart, 1958: Persson et al., 1970) arriving at values that are between 5 and 13 times more than the static.

When the intensity of the stress wave is higher than the dynamic compressive strength RC, a crushing of the rock surrounding the blasthole wall is produced by collapse of the intercrystalline structure. However, this excessive crushing does little to aid in fragmentation and gravely reduces the strain wave energy.

Therefore the following is recommended:

- Explosives that develop blasthole wall strain energy that is lower than or equal to RC must be chosen.
- Provoke a variation in the Pressure-Time curve ($P - t$) by decoupling the charge in the blasthole.

These points are of maximum importance in perimeter or contour blastings.

The powder factors required in bench blastings can be correlated with the compressive strength, as indicated in Table 17.1 (Kutuzov, 1979).

17.2.3 *Porosity*

There are two types of porosity: intergranular or formational, and that of disolution or post-formation.

The first, which has a uniform distribution in the rock mass, provokes two effects:

- Attenuation of the strain wave energy.
- Reduction of the dynamic compressive strength and, consequently, an increase in crushing and percentage of fines.

The fragmentation of very porous rocks is carried out, almost exclusively, by bubble energy, so the following recommendations should be observed:

- Use explosives with a high EB/ET ratio, such as ANFO.
- Increase EB at the cost of ET by decoupling the charges and the initiation systems.
- Maintain the explosion gases at high pressure with an adequate stemming height and type.
- Maintain the burden equal for each hole by using various free faces.

The post-formation porosity is caused by spaces and cavities that result from the disolving of the rock material by underground water (karstification). The empty spaces are much larger and their distribution is much less uniform than in the intergranular porosity.

In rock of volcanic origin it is also frequent to find a large number of cavities formed during its consolidation.

The cavities that are intersected by blastholes not only

Table 17.1. Rock classification according to their facility of fragmentation by explosives in open pit mines.

Powder factor Class limit (kg/m^3)	Average value (kg/m^3)	Mean distance between natural fractures in rock mass (m)	Uniaxial compressive rock strength (MPa)	Rock density (t/m^3)
0.12-0.18	0.150	< 0.10	10-30	1.40-1.80
0.18-0.27	0.225	0.10-0.25	20-45	1.75-2.35
0.27-0.38	0.320	0.20-0.50	30-65	2.25-2.55
0.38-0.52	0.450	0.45-0.75	50-90	2.50-2.80
0.52-0.68	0.600	0.70-1.00	70-120	2.75-2.90
0.68-0.88	0.780	0.95-1.25	110-160	2.85-3.00
0.88-1.10	0.990	1.20-1.50	145-205	2.95-3.20
1.10-1.37	1.235	1.45-1.70	195-250	3.15-3.40
1.37-1.68	1.525	1.65-1.90	235-300	3.35-3.60
1.68-2.03	1.855	> 1.85	> 285	> 3.55

hinder drilling by loss of drill steel and obstruction, but also effect the efficiency of the blast, especially if loosely packed or pumpable explosives are used, Fig. 17.1.

If the boreholes do not intersect the cavities, the yield of the blast also descends because:

– The propagation of radial cracks is interrupted by the cavities.

– The rapid fall in pressure of the gases as the blastholes intercommunicate with the cavities, halting the opening of the radial cracks, while the gases escape towards the empty spaces.

17.2.4 *Internal friction*

As the rocks do not form an elastic media, part of the strain wave energy that propagates through them is converted to heat by diverse mechanisms. These mechanisms are known as *internal friction* or *specific damping capacity, SDC*, which measure the ability of the rock to attenuate strain waves generated by the detonation of the explosive. *SDC* varies considerably with the type of rock: from values of 0.02-0.06 for granites (Windes, 1950; Blair, 1956) up to 0.07-0.33 for sandstones. *SDC* increases with porosity, permeabillity, joints and water content of the rock. It also increases considerably with the meteorized levels in function with their thickness and weathering.

The intensity of the fracturation by the strain wave increases as the *SDC* decreases. Therefore, watergel type explosives are more effective in hard and crystalline formations than in soft and decomposed materials (Cook, 1961; Lang, 1966). On the other hand, in the latter, ANFO is the most adequate even though its strain energy is lower.

17.2.5 *Conductivity*

The leakage or shunting of electrical current can occur when the detonators are placed in blastholes that are in rock of certain conductivity, such as complex sulfides, magnetites, etc., especially when the rocks are abrasive and water is present near the round. The measures that should be taken to avoid these problems are:

– Check that the cables of the detonators are well enclosed in plastic and,

– That all the connections of the circuit are well insulated and protected. It is recommended that rapid connectors be used.

The failure of one of the detonators could considerably affect the results of the blast.

17.2.6 *The composition of the rock and the secondary dust explosions*

The secondary dust explosions usually occur in coal mines and in highly pyritic areas such as underground metal mines, and are more frequent each day due to the use of large diameter blastholes.

The first charges fired create, on one hand, a high quantity of fines which are thrown into the atmosphere and, on the other, agitate the dust deposited on the sidewalls and roof of the excavation with the airblast and vibrations. If the energy of the gases from the last charges is sufficiently high, it could ignite the concentrated dust causing secondary explosions with devastating effects upon the ventilation installations, doors, mobile equipment, etc.

The probability of secondary explosions can be reduced by taking some of the following steps:

– Eliminate the use of aluminized explosives since the particles of Al_2O_3 at high temperatures in the detonation products are potential ignition centers.

– Select an explosive and blasthole geometry for burn cuts which produce coarse material.

– Stem all blastholes with sand, clay plugs or water.

– Create a cloud of limestone or another inhibitor in front of the face by exploding a bag of said material with a detonator fired some miliseconds before the blast.

– Wash the walls and floors of the excavation frequently to remove the deposited dust.

– Fire the blasts after evacuating all personnel from the mine.

17.3 PROPERTIES OF THE ROCK MASS

17.3.1 *Lithology*

The blasts in zones where an abrupt lithological change is produced, for example in waste and ore and, in consequence, a variation in the strength of the rocks, the design must be reconsidered. One of the two following methods could be used:

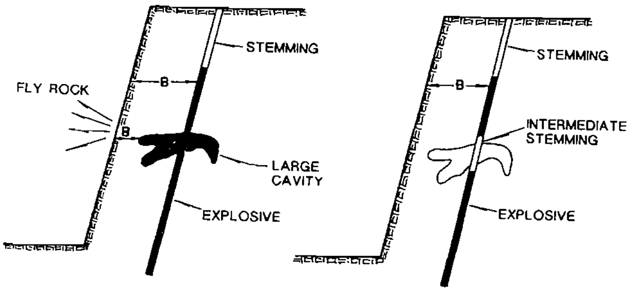


Fig. 17.1. Correct use of a bulk explosive charge in ground with large cavities.

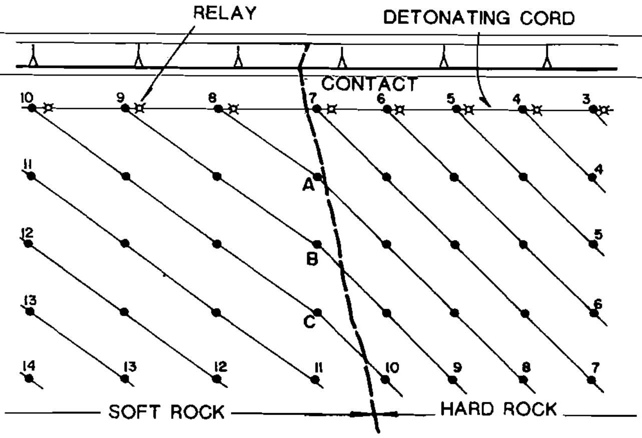


Fig. 17.2. Recommended change in blasthole pattern of *V* type blast at contact between waste and ore.

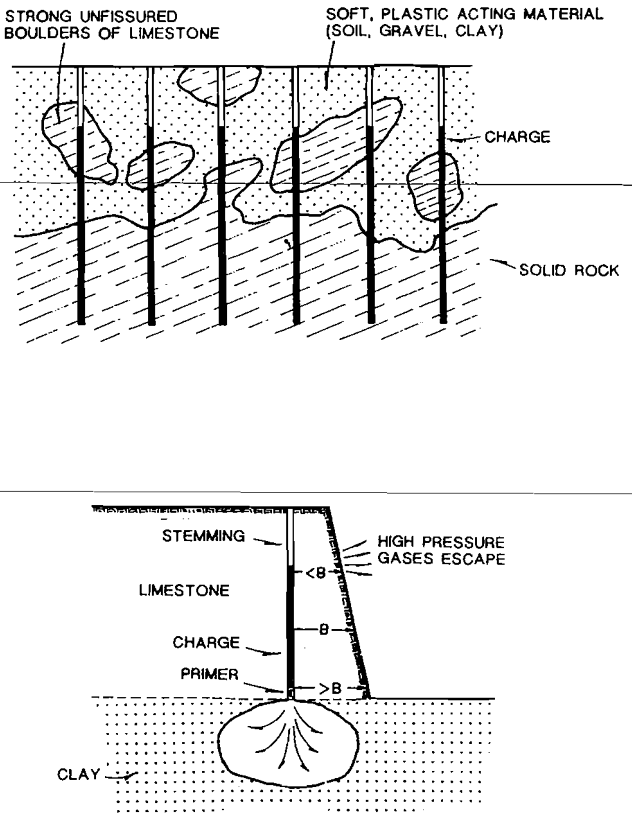

STRONG UNFISSURED BOULDERS OF LIMESTONE
SOFT, PLASTIC ACTING MATERIAL (SOIL, GRAVEL, CLAY)
CHARGE
SOLID ROCK
STEMMING
HIGH PRESSURE GASES ESCAPE
<B
LIMESTONE
B
CHARGE
PRIMER
>B
CLAY

Fig. 17.3. Typical cases of lithological changes with contact between competent rocks and plastic materials (Hagan).

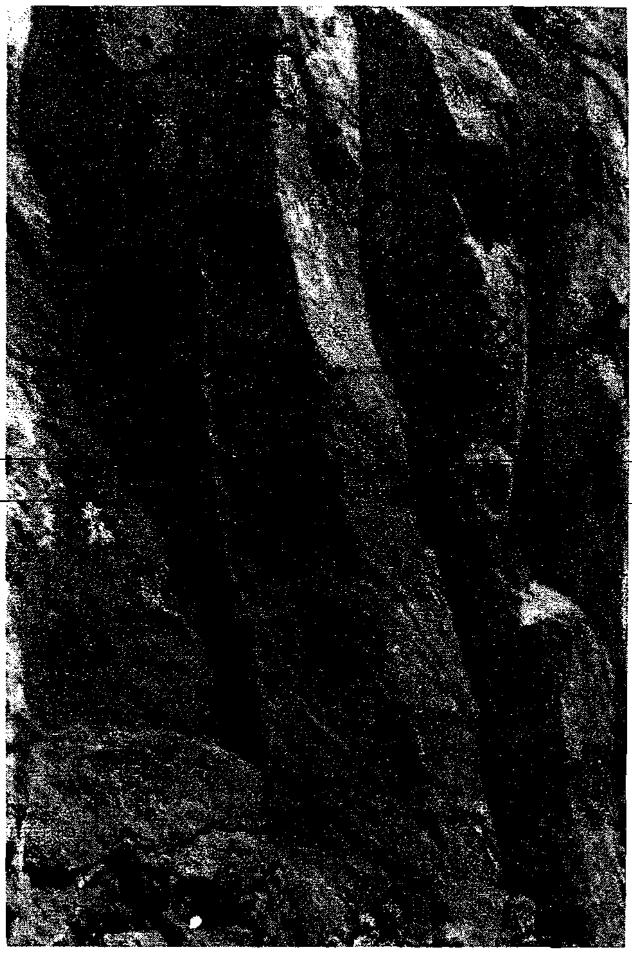


Photo 17.1. Blocks with columnar geometry in basaltic formations.

Photo 17.2. Intensely jointed limestone rock mass.

a) Equal patterns for both types of rock and a variation in the unitary charges.

b) Different patterns with equal charges per hole. This placement is usually adopted maintaining equal burden, Fig. 17.2, as the introduction of a different $S \times B$ pattern for each zone would entail a more complex drilling and the newly created face may be stepped.

The semi-horizontal stratiform beds presented by some very resistant layers may lead to a peculiar type of blastings in which the charges are placed in the blastholes and completely confined at these levels. It is also recommended that the primers of the explosive columns coincide with the strongest levels in order to obtain maximum effect from the strain energy.

Table 17.2.

Absorption of strain wave energy by joints	Joint width (mm)	Nature of joints
1. Small (< 20%)	(A) 0	(A) Tightly stacked
	(B) 0-4.0	(B) Cemented with material of acoustic impedance close to that of the main rock
2. Slight (20-40%)	(A) Up to 0.5	(A) Open joints filled with air or water
	(B) Up to 4.0	(B) Cemented with material of acoustic impedance 1.5-2 times less than that of main rock
3. Medium (40-80%)	0.5-1.0	Open joints filled with air or water
4. Large (> 80%)	(A) 0.1-1.0	(A) Joints filled with loose and porous material
	(B) 1.0	(B) Open joints filled with loose, porous material, air and water

Table 17.3. Possible combinations of spacing between blastholes (S), joints (J), and maximum admissable block size (M).

Case	J_s:S	J_s:M	S:M	Fragmentation sensitive to specific charge	% of boulders
1	$J_s > S$	$J_s > M$	$S > M$	Yes	Medium
2	$J_s > S$	$J_s > M$	$S < M$	Yes	Low
3	$J_s > S$	$J_s < M$	$S < M$	Yes	Low
4	$J_s < S$	$J_s > M$	$S > M$	No	High
5	$J_s < S$	$J_s < M$	$S < M$	No	Low
6	$J_s < S$	$J_s < M$	$S > M$	No	Low

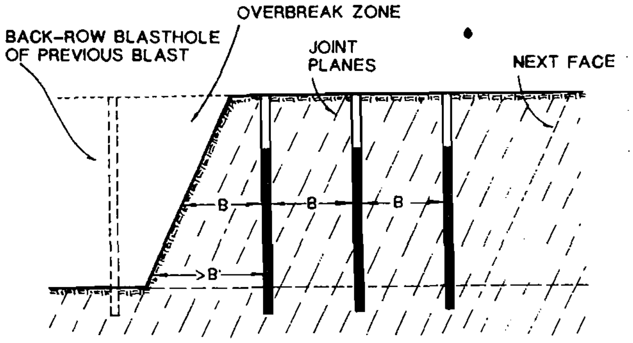


Fig. 17.4. Excessive toe burden caused by structurally controlled backbreak zone and face angle.

When two materials of very different strengths come into contact as, for example, a competent limestone with very plastic clays and, if the blastholes pass through these formations, a great loss of energy associated with a drop in pressure and escape of gases will appear, producing a rapid deformation of soft material and, as a consequence, poor fragmentation, Fig. 17.3.

In order to increase the yield of the blasts in these cases, the following is recommended:

- Stem with adequate material the zones of the blastholes that are in contact with or near plastic material.
- Use explosive charges that are totally coupled to the competent rock with a high detonation velocity and *ET/EB* relationship.
- Place the primers in the middle of the hard rock to increase the resulting strain wave that acts upon both sides.
- Avoid premature escape of gases to the atmosphere insuring that both the stemming height (at least 20 D) and the size of the burden are correct at the top of the blastholes.

17.3.2 *Pre-existing fractures*

All rocks in nature have some type of discontinuity, microfissures and macrofissures, which decisively influence the physical and mechanical properties of the rocks and, consequently, the blasting results, Photos 17.1 and 17.2.

The areas of discontinuity can be varied: bedding planes, planes of lamination and primary foliation, planes of schistose and slate, fractures and joints.

The discontinuities can be tight, open or filled and, for this reason, can exhibit different degrees of explosive energy transmission. Table 17.2. The walls of these discontinuities are even surfaces upon which the strain waves may be reflected, suffering attenuation and dispersion.

The fragmentation is influenced by the spacing between blastholes *S*, the separation between joints *J* and the maximum admissible block size *M*. In Table 17.3, various possible combinations are indicated, as well as their repercussion upon the percentage of forseen boulders.

Another aspect of the design of the blastings is referred to as geostructural control of the rock mass, which refers to the relative orientation of the face and break direction of the round with respect to the strike and dip of the strata. In Table 17.4, the forseen results for the different cases that might arise are indicated, taking into account the inclination of the discontinuitites and the relative angle of the strike and dip.

Special precautions should be taken when the discontinuities are subvertical and the direction of the shot is normal (parallel) to theirs, because overbreak is frequent behind the last row of blastholes and inclined drilling becomes necessary to maintain the burden dimension in the first row of the round. Fig. 17.4 and Photo 17.3.

When the bedding or the joint system has an angle smaller than 30°, it is recommended that the blastholes be normal to said planes in order to increase the yield of the blasts.

In tunnel excavations, the structural characteristics largely condition the geometry of their profile, almost rectangular if the rocks are massive and with a curved arch if the rock is more unstable. When the discontinuities are normal to the tunnel's axis, the blasts usually have good results. Fig. 17.5a. If the bedding or the discontinuities are parallel to the axes of the tunnels, Fig. 17.5b, frequently the advances are not satisfactory and the faces are uneven. When the bedding has an oblique direction with respect to the axis of the tunnel, there will be one

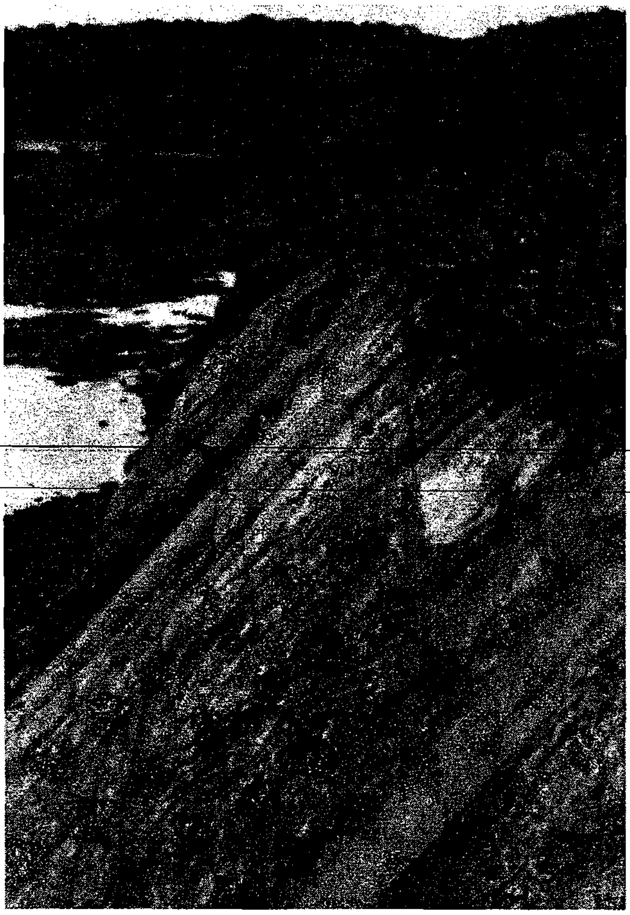


Photo 17.3. Face of a blast that coincides with a bedding plane.

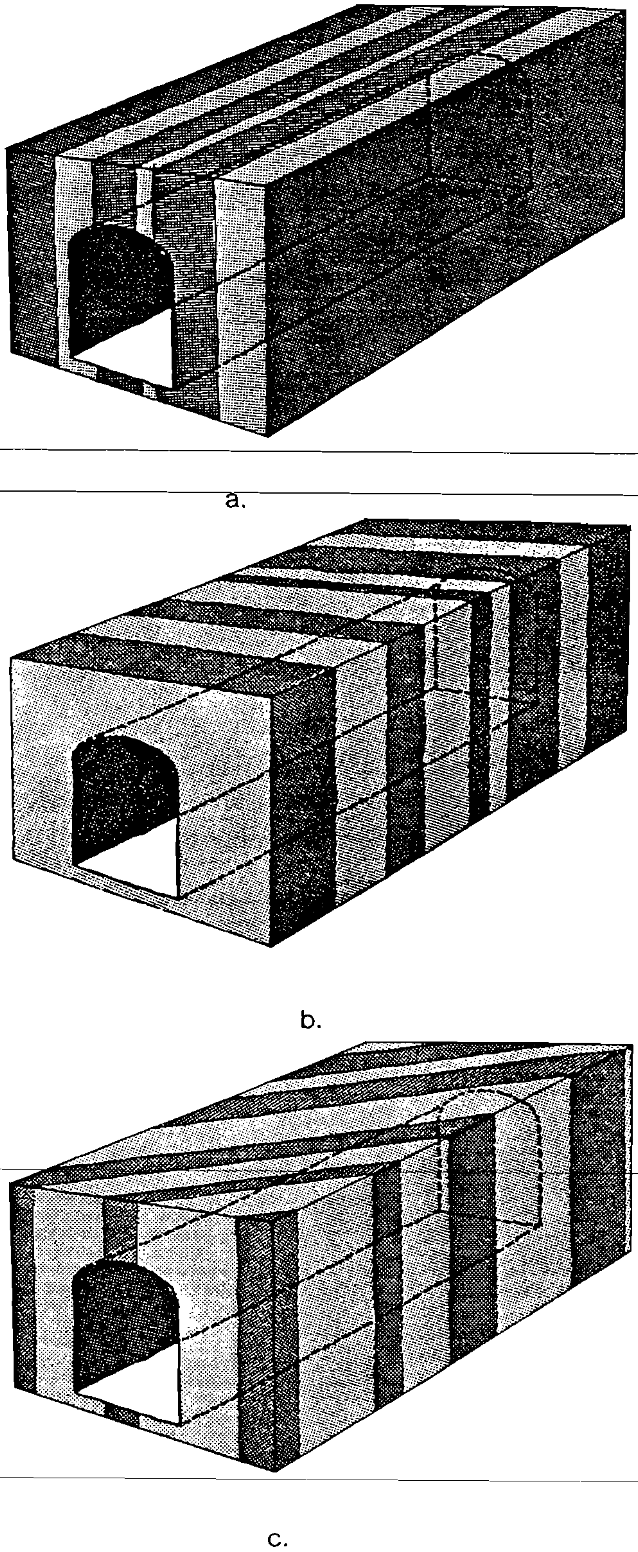


Fig. 17.5. Relative directions of the beds with regard to the axes of the tunnels.

side on which it is easier to blast, such as in the case of Fig. 17.5c, the left side. On the other hand, very laminated rocks with high schistosity and fissurization respond well to V cuts, and in large diameter tunnels very deep pulls of up to 6 m are possible with this type of cut.

When V cuts are used in sinking rectangular shafts, the best results are obtained when the discontinuities are parallel to the line joining the bottom of the V cut, Fig. 17.6.

17.3.3 *Stress fields*

When the stress fields, either tectonic and/or gravitational (non-hydrostatic) act, the fracture pattern generated around the blastholes can be influenced by the non-uniform stress concentrations around the same.

In homogeneous massive rock, the cracks which start to propagate radially from the blastholes tend to follow the direction of the principal stresses.

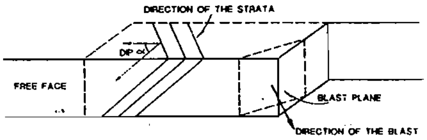
Therefore, when driving shafts in rock masses with a high concentration of residual stresses, as in the case of Fig. 17.7, the firing sequence in the blastholes of the cut should be adapted to this situation.

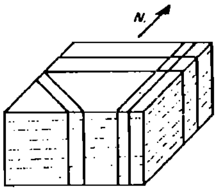
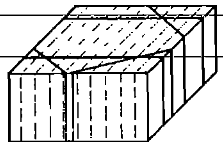
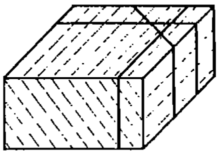
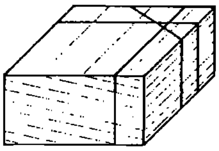
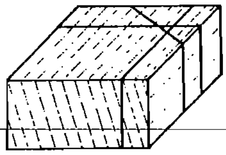
If in the presplitting planes of the planned excavation the influencing stresses are normal to the same, the obtained results will not be satisfactory unless the spacing is considerably reduced or a pilot excavation is carried out to relax the mass and free the stresses, and presplitting is substituted for smooth blasting.

17.3.4 *Water content*

Porous and intensely fissured rock, when saturated with water, usually pose certain problems:

Table 17.4. Design of the blasts with attention to geostructural control.



	Inclination of the strata	Angle between the direction of the strata and the blast break	
	α = 0°	Indifferent break direction	
	α = 90°	β = 0° = 180° = 360°	Good fragmentation, irregular face
		β = 45° = 135° = 225° = 315°	Variable fragmentation, saw-tooth face
		β = 90° = 270°	Most favorable direction
	α = 45°	β = 0° = 180° = 360°	Good
		β = 45° = 135°	Unfavorable
		β = 90°	Not very favorable
		β = 225° = 315°	Acceptable
		β = 270°	Very favorable
	0° < α < 45°	β = 0° = 180° = 360°	Good
		β = 45° = 135°	Unfavorable
		β = 90°	Not very favorable
	(Similar to the previous case hardness is determining factor)	β = 225° = 315°	Acceptable
		β = 270°	Very favorable
	45° < α < 90°	β = 90°	Not very favorable
		β = 270°	Favorable
		(Depending upon the value of α and upon the rock competence, the results will be closer a α = 45° ó α = 90°)	

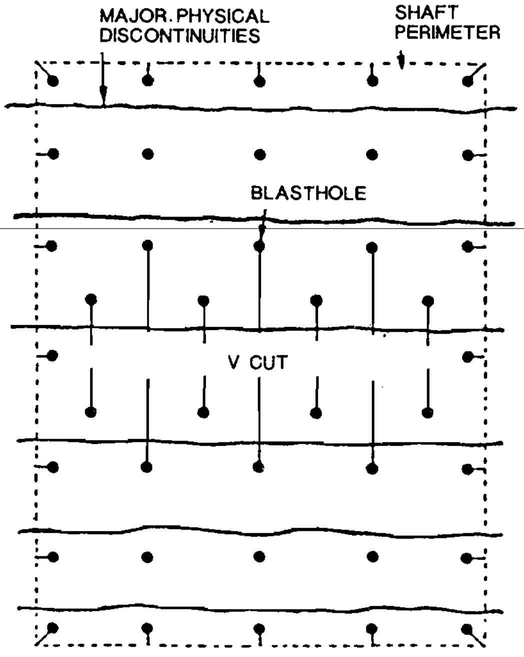


Fig. 17.6. Rectangular sinking shaft with V cut. (Hagan, 1983)

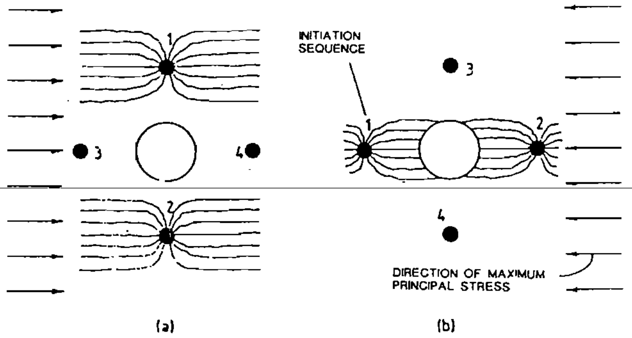


Fig. 17.7. Initiation sequence for burn cut in high horizontal stress field: (a) to be avoided, (b) satisfactory. (Hagan, 1983).

– Only explosives that are unaltered by water can be used.

– Blastholes are lost due to caving, and

– Inclined drilling is difficult.

On the other hand, water affects the rock and the rock masses by the following:

– Increase in propagation velocity of the elastic waves in porous and fissured ground.

– Reduction of the compressive and tensile strength of the rocks (Obert and Duvall, 1967) as the friction between particles is lower.

– Reduction of the stress wave attenuation and, because of this, the breakage effects are intensified by *ET* (Ash, 1968).

– Water filled joints allow the passage of strain waves without internal spalling. But when the mass enters in tension, the water is mobilized, forming a wedge which could provoke a great overbreak.

17.3.5 *Temperature of the rock mass*

The orebeds that contain pyrites usually have high rock temperature problems because of the effect of slow oxidation of this mineral, causing the explosive agents such as ANFO to react exothermically with the pyrite, with stimulation from 120°C ± 10°C.

The latest investigations point to a first reaction between ANFO and hydrated ferrous sulphate, and even more so between the latter and ammonic nitrate, initiating an exothermic reaction that is self-maintaining from 80°C on. This ferrous sulphate is one of the products of decomposition of the pyrites, apart from the ferric sulphate and the sulfuric acid.

To avoid this problem, which has caused severe accidents on several occasions, diverse substances which inhibit ANFO have been added, such as urea, potassic oxalate, etc., arriving at the conclusion that by adding to ANFO a 5% in weight of urea, the exothermic reaction of the ternary mixture is avoided up to a temperature of 180°C (Miron et al., 1979).

The sensitivity of the water gel type explosives also depends highly upon the temperature of the rock with which it is in contact and, because of this, great attention must be paid to this phenomenon.

A general recommendation when these problems are present is to limit the number of blastholes per blast, in order to lower the time that passes between the charging and the firing.

REFERENCES

Ash, R. I.: *The design of blasting rounds.* Ch. 7.3. Surface Mining, Ed. E. P. Pfleider, AIME, 1968.

Atchison, T.C.: *Fragmentation principles.* Ch. 7.2. Surface Mining, Ed. E. P. Pfleider, AIME, 1968.

Belland, J. M.: *Structure as a control in rock fragmentation. Carol Lake iron ore deposits.* CIM Bulletin, March 1966.

Bhandari, S.: *Blasting in non-homogeneous rocks.* Australian Mining, May, 1974.

Blair, B.E.: *Physical properties of mine rock.* Part III. USBM RI No. 5130, 1955: Part IV USBM-RI, No. 5244, 1956.

Grant, C. H.: *How to make explosives do more work.* Mining Magazine, August, 1970.

Hagan, T. N.: *The effects of some structural properties of rock on the design and results of blasting.* ICI Australia Operations PTY. Ltd. Melbourne, 1979.

Hagan, T. N.: 'The influence of rock properties of blasts in underground construction. Proc. Int. Symp. on Engineering Geology and Underground Construction. Lisboa, Portugal, 1983.

Harries, G.: *Breakage of rock by explosives.* Aus. I. M. M., London, 1978.

Kutuzov, B. N. et al.: *Classification des roches d'apres leur explosibilité pour les decouvertes.* Gornyl, Zurnal, Moscow, 1979.

Lopez Jimeno, E.: *Influencia de las propiedades de las rocas y Macizos Rocosos en el diseño y resultado de las voladuras.* Tecniterrae, 1982.

Merrit, A. H.: *Geological predictions for underground excavations.* North American RETC Conference.

Polak, E.J.: *Seismic attenuation in engineering site investigations.* Proc. Ist. Aust. N.Z. Conf. Geomechanics, Melbourne, 1971.

Rinehart, J.S.: *Fractures and strain generated in joints and layered rock masses by explosions.* Proc. Symp. Mechanism of Rock Failure by Explosions. Fontainebleau, October, 1970.

Sassa, K. & I. Ito: *On the relation between the strength of a rock and the pattern of breakage by blasting.* Proc. 3rd. Congress International Society of Rock Mechanics, Denver, 1974.

Sjogren, B. et al.: *Seismic classification of rock mass qualities.* Geophysical Prospecting, No, 27, 1979.

Wild, H.W.: *Geology and blasting in open pits.* Erzmetall, 1976.

CHAPTER 18

Characterization of the rock masses for blast designing

18.1 INTRODUCTION

The properties of rock masses that most influence blast design are:

- Dynamic strengths of the rocks.
- Spacing and orientation of the planes of weakness.
- Lithology and thickness of the sedimentary bedding planes.
- Velocity of wave propagation.
- Elastic properties of the rocks.
- Types of infilling material and tightness of the joints.
- Indexes of anisotropy and heterogeneity of the rock masses, etc.

The determination of these parameters by direct or laboratory methods is very costly and difficult, as the samples tested do not usually include discontinuities and the lithological changes of the rock mass from where they were taken. In order to obtain a representative sample, it would be necessary for it to have a size ten times larger than the mean spacing between joints. However, these methods do complement the characterization of the rock masses to be blasted.

At the moment, the most common geomechanic techniques for monitoring are:

- Diamond drilling with core recovery and geomechanic testing.
- Structural studies of the joint system.
- Seismic survey profiles.
- Geophysical logs of investigation drill holes.
- Geophysical logs of production blastholes.
- Logging and individual treatment during drilling of production blastholes.

18.2 DIAMOND DRILLING WITH CORE RECOVERY AND GEOMECHANIC TESTING

With core recovery by diamond drilling, one of the most extensive rock mass clasifications known can be applied, called RQD (Rock Quality Designation, Deere 1968) which is defined as the percentage of the core length recovered in pieces larger than 10 cm with respect to the length of the core run, Table 18.1.

Apart from this, the geomechanic testing of Point Load Strength I_s can be carried out either in the diametral or axial position, to be able to estimate the Uniaxial Compressive Strength RC.

$$RC\,(\text{MPa}) \simeq 24 \cdot I_s\,(50)\,(\text{MPa})$$

Borquez (1981) determines the blastability factor K_v from the Pierce equation, for the calculation of the Burden from the RQD index, corrected by a Coefficient of Alteration which takes into account the Joint Strength as a function of their tightness and the type of infilling, Fig. 18.1 and Table 18.2.

The company Steffen, Robertson and Kirsten, Ltd. (1985), used various geomechanic parameters to calculate the powder factors in bench blasting, among which RQD, the Uniaxial Compressive Strength (MPa), the Internal Friction Angles and Abrasiveness of the joints and the Density are found (t/m^3), Fig. 18.2. This procedure is one of the few that take into account the effect of blasthole diameter (mm) or spacial distribution of the explosives upon the powder factor of the blast.

18.3 CHARACTERISTICS OF THE JOINT SYSTEMS

There are various properties of the joints that can be measured in a characterization study, but the most important from a breakage point of view are spacing and orientation.

An index obtained frequently is that known as the *Volumetric Joint Count*, J_v which is defined by the total number of joints per cubic meter, obtained from the summing of the joints present per meter for each one of the existing families.

The relationship between the index J_v and the RQD is, according to Pallsmtrom (1974), the following:

$$RQD = 115 - 3.3\,J_v \text{ For } J_v < 4.5,\ RQD = 100$$

According to the orientation of the joints, the in-situ blocks will show different geometries that doubly affect the fragmentation of the blast and the most useful break direction of the round.

In Fig. 18.3, the approximate volume of the blocks taken from J_v and the relationship of the three characteristic intersections of the same can be estimated.

An attempt to take into consideration the structural discontinuities when designing the rounds is owed to Ashby (1977), which relates the fracture frequency and their shear strength to the powder factors of the explosive, Fig. 18.4.

Lilly (1986) defined a Blastability Index BI that is

Table 18.1.

RQD	Rock quality
0–25	Very poor
25–50	Poor
50–75	Fair
75–90	Good
90–100	Excellent

Table 18.2.

Joint strength	Correction factor
Strong	1.0
Medium	0.9
Weak	0.8
Very weak	0.7

Table 18.3.

J_v	Characteristics of the mass
> 1	Massive blocks
1–3	Large blocks
3–10	Medium size blocks
10–30	Small blocks
> 30	Very small blocks

Table 18.4.

Geomechanic parameters	Rating
1. Rock mass description (RMD)	
1.1 Powdery/friable	10
1.2 Blocky	20
1.3 Totally massive	50
2. Joint Plane Spacing (JPS)	
2.1 Close (< 0.1 m)	10
2.2 Intermediate (0.1 to 1 m)	20
2.3 Wide (> 1 m)	50
3. Joint Plane Orientation (JPO)	
3.1 Horizontal	10
3.2 Dip out of face	20
3.3 Strike normal to face	30
3.4 Dip into face	40
4. Specific Gravity Influence (SGI) SGI = 25 SG – 50 (where SG is in Tons/cu metre)	
5. Hardness (H)	1–10

obtained by summing the representative values of five geomechanics parameters.

$$BI = 0.5\,(RMD + JPS + JPO + SGI + HD)$$

In Table 18.4, the ratings for Blastability Index parameters are described.

The Powder Factors *CE* or the Energy Factors *FE* are calculated with Fig. 18.5, or the equations

$$CE\ (\text{Kg ANFO/t}) = 0.004 \times BI, \text{ or}$$

$$FE\ (\text{MJ/t}) = 0.015 \times BI$$

From the numerous experiences carried out in Australia, it has been concluded that the Rock Factor of the Model Kuz-Ram of Cunningham (1983) can be obtained by multiplying *BI* by 0.12.

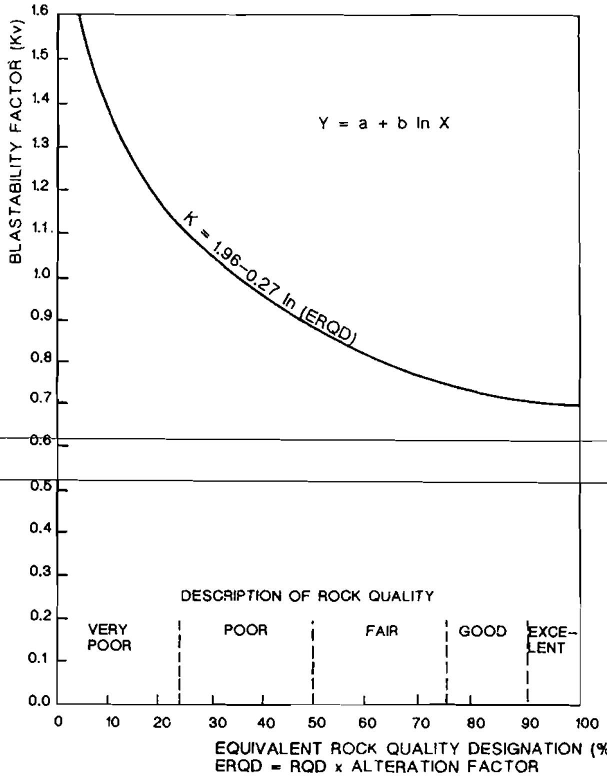


Fig. 18.1. Blastability factor k_v vs equivalent rock quality designation, RQDE.

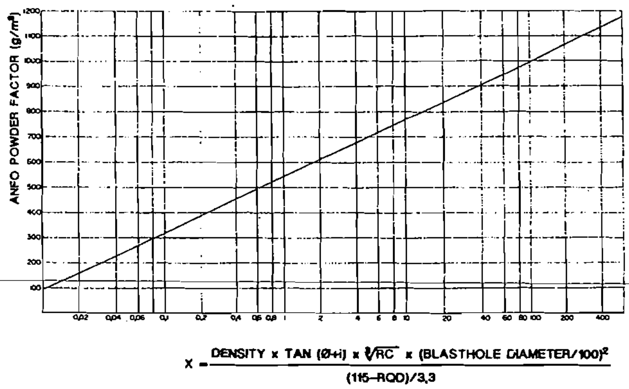


Fig. 18.2. Calculation of the Powder Factor as a function of the different geomechanic parameters of the rock mass.

Example:

Consider a highly laminated, soft ferruginous shale which has horizontal to sub-horizontal bedding to which the following values correspond:

RMD = 15
JPS = 10
JPO = 10
SGI = 10
H = 1

The total sum is 46 and the Blastability Index is *BI* = 23.

From Fig. 18.4, a powder factor of 0.1 kg/t is obtained.

Ghose (1988) also proposes a geomechanic classification system of the rock masses in coal mines for predicting powder factors in surface blastings. The four parameters measured are indicated in Table 18.5.

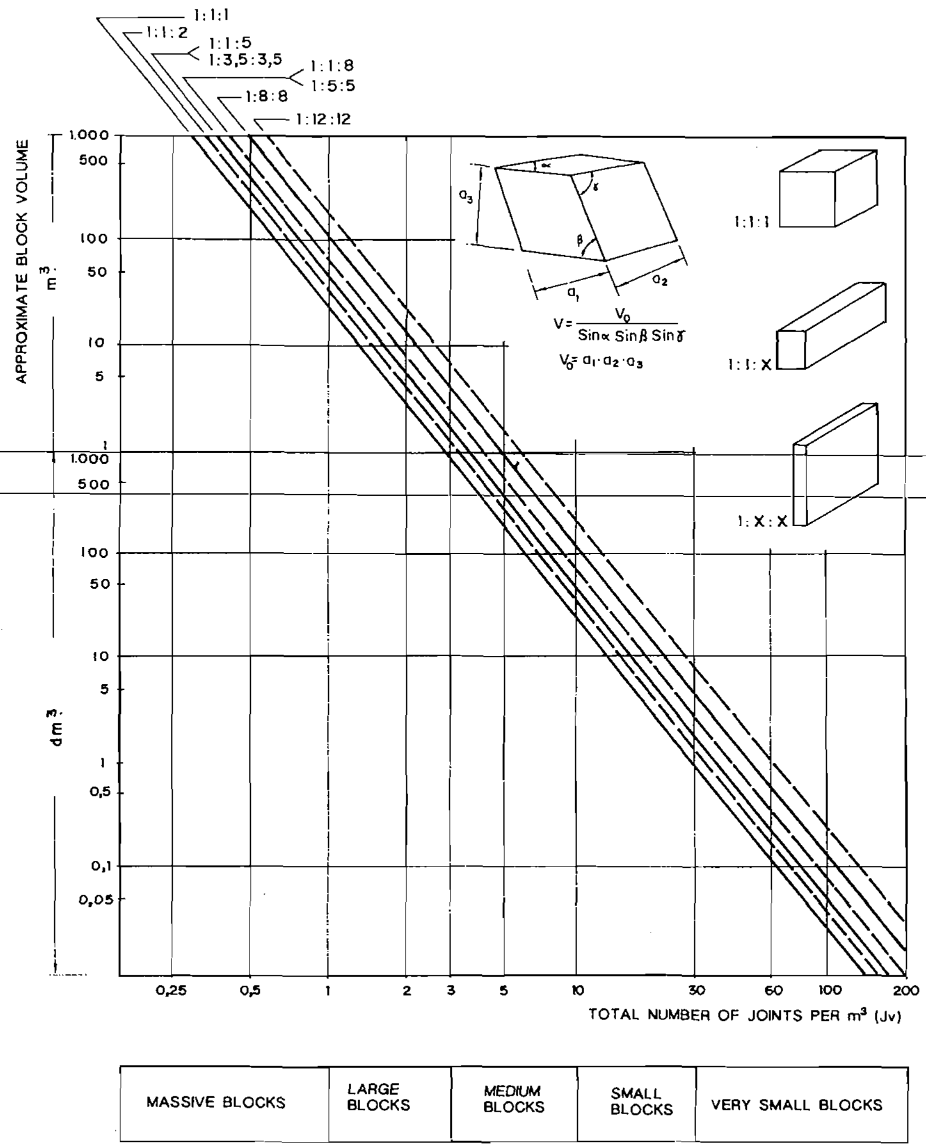


Fig. 18.3. Estimation of the volume of the in-situ blocks.

Table 18.5.

Parameters	Range of values				
1. Density	1.3–1.6	1.6–2.0	2.0–2.3	2.3–2.5	> 2.5
Ratio	20	15	12	6	4
2. Spacing of discontinuities (m)	< 0.2	0.2–0.4	0.4–0.6	0.6–2.0	> 2.0
Ratio	35	25	20	12	8
3. Point load strength Index (MPa)	< 1	1–2	2–4	4–6	> 6
Ratio	25	20	15	8	5
4. Joint plane orientation	Dip in to face	Strike at an acute angle to face	Strike normal to face	Dip out of face	Horizontal
Ratio	20	15	12	10	6

Table 18.6.

Adjustment factors	Values
1. Degree of confinement	
Highly confined	– 5
Reasonably free	0
2. Bench stiffness	
Hole depth/burden > 2	0
Hole depth/burden < 1.5	– 5
Hole depth/burden 1.5–2	– 2

Table 18.7.

Blastability index	Powder factor (kg/m³)
80–85	0.2–0.3
60–70	0.3–0.5
50–60	0.5–0.6
40–50	0.6–0.7
30–40	0.7–0.8

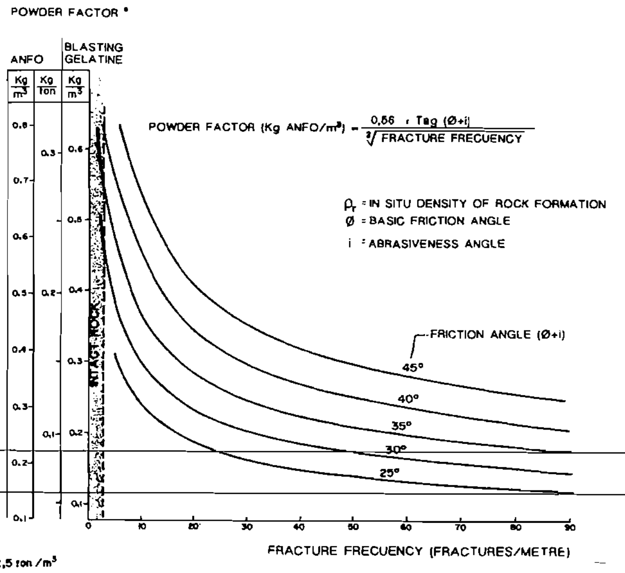


Fig. 18.4. Empirical relationship between powder factor, fracture frequency and joint shear strength.

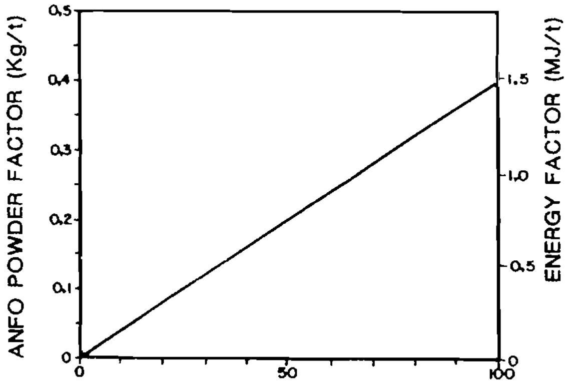


Fig. 18.5. Calculation of the Powder Factors or Energy Factors from the Blastability Index (Lilly, 1986).

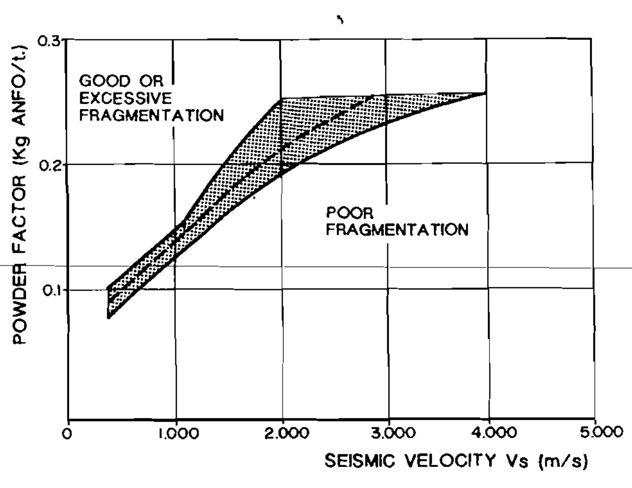


Fig. 18.6. Correlation between the seismic velocity and the powder factor.

The value obtained is adjusted to take into account the conditions under which the blast is carried out. In Table 18.6, the different values used are indicated.

From experiences carried out in 12 surface mines, the correlation between the blastability indexes and the powder factors were established, using a watergel as the pattern explosive with a detonation velocity of 3.800 m/s.

18.4 SEISMIC SURVEY

The first applications of the seismic survey in the design of blasting was carried out by Broadbent (1974), Heynen and Dimock, who relationed the powder factor with the seismic propogation velocity, Fig. 18.6.

As can be observed, when the velocity increases, a larger quantity of energy is required to obtain satisfactory fragmentation. It is well known that the criterium for equality of impedances (Rock propogation velocity × rock density = Detonation velocity × explosive density) with the purpose of maximizing the transfer of the explosive energy to the rock.

This method has had great success in several operations where drilling and blasting costs have been reduced by up to 15 %.

18.5 GEOPHYSICAL TECHNIQUES TO OBTAIN ROCK MASS DATA

Exploration drillholes, with or without core recovery, for geophysical logging has the following disadvantages:

- It is costly and time consuming and,
- It requires special drilling and logging equipment.

For these reasons it is not a usual operational procedure, except in areas where large installations are to be built, such as power plants, storage depots, etc., or in those cases where the aparatus is free and can be directed to other uses, for example rock breakage with explosives.

18.6 LOGGING OF PRODUCTION BLASTHOLES

This procedure is relatively simple, fast and trustworthy, as it studies the whole of the blast and only requires investment in logging equipment.

The technological advances made in the production of logging aparatus allow the user to obtain the following information:

- The location of weak bedding such as coal seams or weathered material.
- Variations in rock strength, and
- The spacing of joints and planes of discontinuities.

The most common logging methods are:

- Sonic logs
- Density logs
- Natural Gamma logs
- Caliper logs

In Fig. 18.7, the data obtained from an overburden with

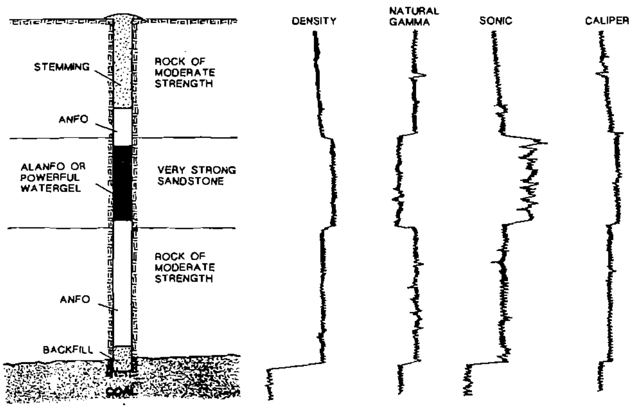

STEMMING
ANFO
ALANFO OR POWERFUL WATERGEL
ANFO
BACKFILL
ROCK OF MODERATE STRENGTH
VERY STRONG SANDSTONE
ROCK OF MODERATE STRENGTH
DENSITY
NATURAL GAMMA
SONIC
CALIPER

Fig. 18.7. Typical geophysical logs and appropriate charge distribution in overburden comprising rocks of moderate and very high strength.

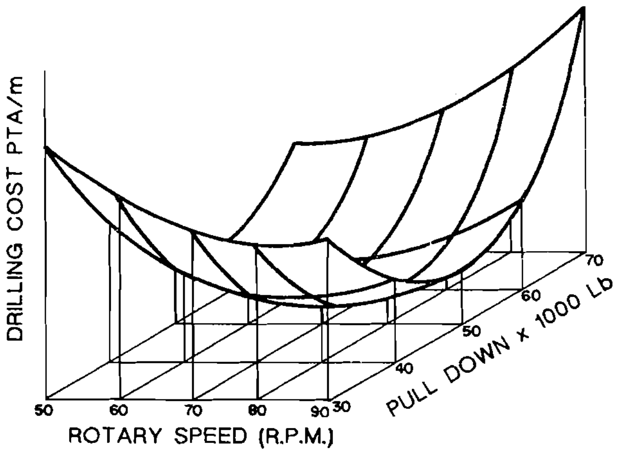


Fig. 18.8. Effect of pulldown force and rotary velocity upon the cost of drilling.

Table 18.8.

Sonic velocity (m/s)	Excavation characteristics
< 1.500	Strata excavatable by scrapers, large draglines, shovels or bucket wheel excavators, without blasting.
1.500–2.000	Easy rippping. Slow difficult digging of unblasted strata by dragline, shovel or bucketwheel excavator.
2.000–2.500	Slow costly ripping. Light blasting (i.e. large blasthole patterns, long stemming columns, low powder factors) may be necessary for large draglines, shovels or bucket wheel excavators.
2.500–3.000	Light blasting is required.
> 4.500	Heavy blasting is required (i.e., small blasthole patterns, short stemming columns, high powder factors).

beds of stronger more massive rocks in between can be observed.

At the present time, there is little available data to correlate the values obtained from the diagraphs with the excavation characteristics. However, Hagan and Gibson (1983) established, based upon their own experience, the classification given in Table 18.8.

18.7 CHARACTERIZATION OF THE ROCK MASS DURING BLASTHOLE DRILLING

Currently, aparatus has been developed to monitor the performance of production drills. For example, the Empasol system, manufactured by the French firm Soletanche, the American G.L.I., etc.

By using these systems one can:

- Evaluate the efficiency of the drilling equipment and method of drilling used.
- Use the information as an aid in mine planning.
- Detect drill failures and inadequate handling of the machinery, and
- Use this as an investigation tool not only to optimize drilling, Fig. 18.8, but also to detect small variations in the rock properties.

This system is considered the most interesting as the investment is minimum and it allows the monitoring of data during the actual drilling.

The recorders can monitor the diverse parameters among which the following are emphasized:

- Air pressure.
- Rotary torque.
- Pulldown rate on the drill bit.
- Rotary revolutions per minute.
- Instantaneous penetration rate.
- Vibrations of the mast.
- Retention force of the drill string.
- Acceleration produced by reflected ground energy, and
- Drilling time.

The recorded values give a complete image of the ground's data. Some of the most used indexes at present time are:

a) *The index of rotation energy*

$$IE = \frac{T_r \times N_r}{VP}$$

where: T_r = Rotary torque, N_r = Rotary speed, VP = Penetration rate.

b) *Degree of alteration index*

$$IA = 1 + \frac{E}{E_o} - \frac{VP}{VP_o}$$

where: E = Pulldown weight on the drill bit, VP = Penetration rate, E_o and VP_o = Maximum values of E and VP.

c) *Index of ground strengths for drilling*

$$IR = E \times \frac{N_r}{VP}$$

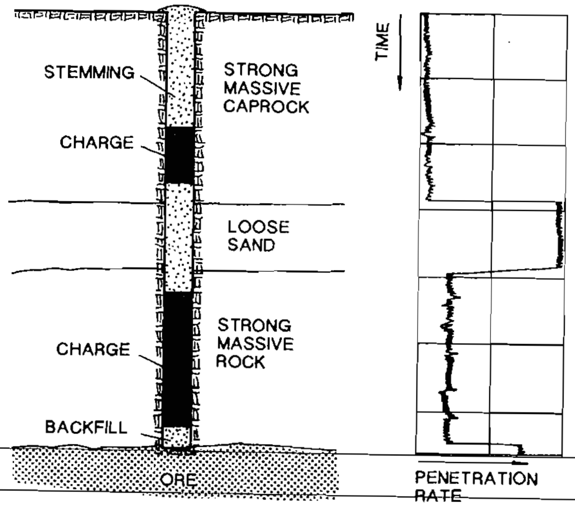


Fig. 18.9. Overburden consisting of adjacent beds with highly dissimilar strengths (Hagan and Reid).

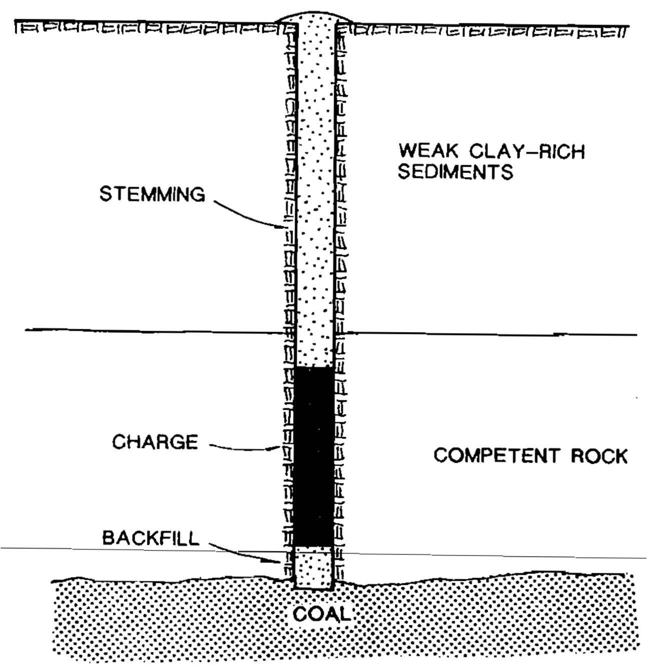


Fig. 18.10. Charging of blasthole which passes through thick weak upper strata (Hagan and Reid).

where: E = Pulldown weight on the drill bit, N_r = Rotary speed, VP = Penetration rate.

The most important parameters are penetration rate and rotary torque. In rocks with high compressive strengths, penetration rates are comparatively low and the rotary torques quite high, unless the discontinuitiy spacings are small in comparison with the blasthole diameter.

When a bed of sand, clay, highly weathered or fissured rock is drilled through, the penetration rate will increase and the required rotary torque will be low, provided that the flow of air is sufficient to flush the cuttings from the blasthole. The pulldown and rotary torque will be combined to obtain maximum performance.

When drilling beds of highly dissimilar strengths, pronounced variations in the penetration rate are observed, Fig. 18.9.

This record would reflect:

- The relative ease with which the rock will be fragmented during the blasting, and
- The correct distribution of the explosive for optimum results.

In the following, the fields of application for this technique will be analyzed for different types of beds.

18.7.1 *Surface coal mines*

In coal seams, the overburden usually consists of strata that have highly variable strengths and, for this reason, the monitoring of blasthole drilling has a bright future.

The data obtained from the diagrams are:

- The thicknesses of the beds which have different strengths.
- The exact depth of the top and wall of the coal seam.

When competent strata lies below a highly weathered area of the same material or an unconsolidated sediment, it will only be necessary to charge the part of the blasthole below contact, Fig. 18.10.

Where a thick layer of soft or highly deformable material, such as sand, lies between beds of competent rock and a continuous charge is placed along the length of the blasthole, the following will occur:

- The gases will rapidly expand towards the deformable area, and
- The rapid fall of the gas pressure in the competent bed will cause poor fragmentation, little swelling and reduced displacement of the muckpile.

The placing of a stemming in the weak level will avoid this and, above all, waste of explosion energy, Fig. 18.11.

18.7.2 *Metalliferous open pits*

In this type of operations, the following cases can be found:

a) *Blastings in the ore/waste contact*

In Fig. 18.12, a blast block containing medium strength waste, weathered ore and high strength ore.

In such a complex case it is possible to modify the drilling pattern, but it would be necessary to have knowledge of the lines of contact before replanning the blast. The most suitable approach consists in standardizing the drilling pattern and modifying the charging of blastholes according to a recording of the penetration rate as indicated in the Fig. 18.13.

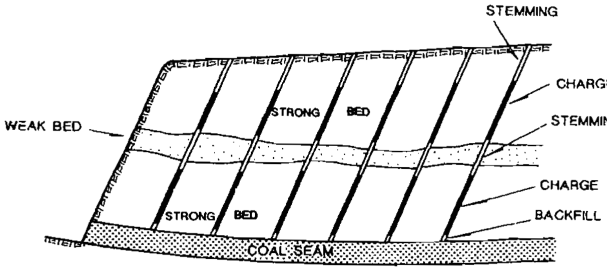


Fig. 18.11. Location of the top of the coal seam and the use of intermediate stemming at the level of a weak in between bed.

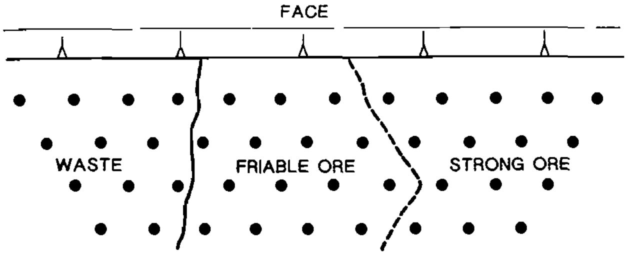


Fig. 18.12. Blast block in an open pit containing highly variable strata (Hagan and Reid).

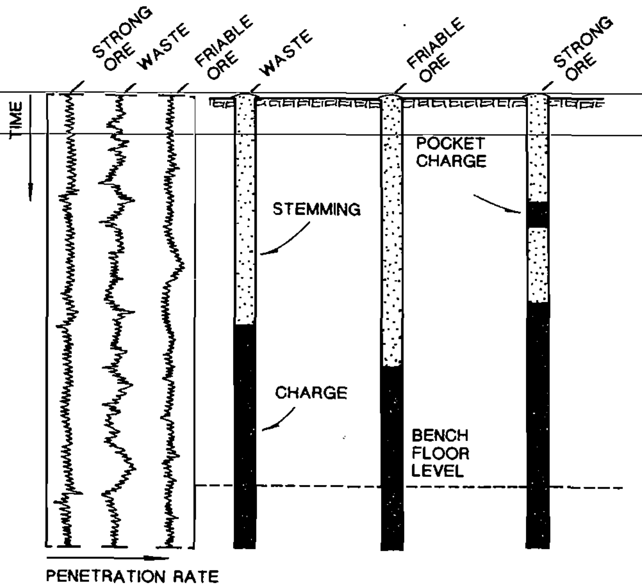


Fig. 18.13. Typical penetration rate-time recordings and charge distributions for effective rock types shown in Fig. 18.12. (Hagan and Reid).

The use of this system brings about the following advantages:

– It avoids an excessive waste of explosive in weak formations.

– Increases the fragmentation yield per unit of weight of explosive.

– Increases control over the disturbing effects of the blast such as vibrations, air blast, fly rock, overbreak, unstable remaining walls, etc.

b) *Boulders within a softer matrix*

If boulders are bedded within a soft, plastic type matrix such as clay, the strain wave created in the blasting is greatly attenuated in these materials.

When a blasthole passes through a floater and has no explosive in that area, the boulder will remain intact and appear as such in the muckpile, Photo 18.1, where it will hinder digging operations and will require secondary blasting.

By continuous chart recorders of the drilling, the blastholes which pass through the boulders would be known and the depths of entrance and exit of these blocks could be determined, establishing selective deck charging of the explosive, Fig. 18.14.

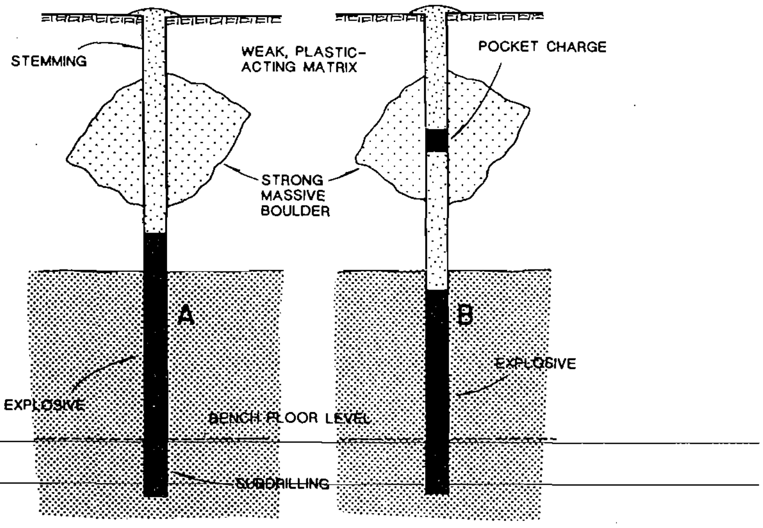


Fig. 18.14. Breakage of strata containing floaters.

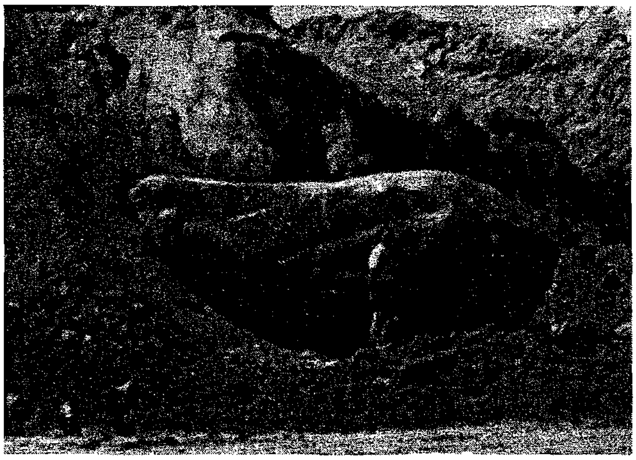


Photo 18.1. Granite boulder within a clay matrix in Meirama.

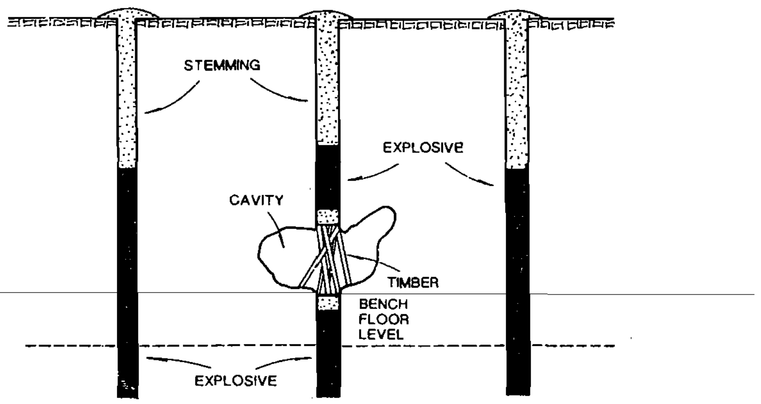


Fig. 18.15. Charging in and around a blasthole containing a large cavity (Hagan and Reid).

c) *Voids or cavities*

The voids are produced by dissolution of primary rock caused by underground water or from another process. Some iron ores and limestones contain cavities of up to 15 meters, with a random distribution.

The continuous drill chart recorders allow:

- The location of blastholes which pass through cavities.
- Knowledge of the depths of entry and exit of the voids.

And afterwards:

- The proper charging of the blastholes which intersect the cavities with deck charges.
- The charging of adjacent blastholes with high strength explosives to compensate for the loss of energy due to the mentioned cavities.

18.8 THE ATTEMPT TO CORRELATE DRILLING INDEXES WITH THE BLASTING DESIGN PARAMETERS

Taking into account that the drilling of a rock constitutes a process of breakage of its structure in which numerous geomechanic factors have influence, it seems logical that the designing of the blastings should be based upon upon the drilling indexes.

In this sense, the following investigations have been carried out:

- Praillet (1980)
- Leighton (1982) with the RQI index.
- Lopez Jimeno (1984) with the Ip index.

18.8.1 *Praillet*

R. Praillet calculates the compressive strength of the rock from the penetration rate, pulldown weight, rotary speed and diameter. Afterwards, by using a third degree equation, he determines the burden value as a function of:

- Bench height.
- Charge density.
- Detonation velocity.
- Stemming height.
- Compressive strength.
- Constant that depends upon the type of loading equipment used: rope shovel or dragline.

The advantage of this system is that it calculates the drilling pattern as function of parameters known beforehand, with the exception of the compressive strength which must be estimated from previous data.

On the other hand, its disadvantage is that as the compressive strength must be determined from the drilling parameters, the system cannot be established until after a few blastholes have been drilled, making the method valid only for very homogeneous formations.

18.8.2 *RQI Index*

Mathis (1975) suggested an index which he called 'RQI' (Rock Quality Index):

$$\mathrm{RQI} = E_h \frac{t}{L}$$

where: E_h = Hydraulic pressure of the drill, t = Drilling time of the blasthole, L = Length of the blasthole.

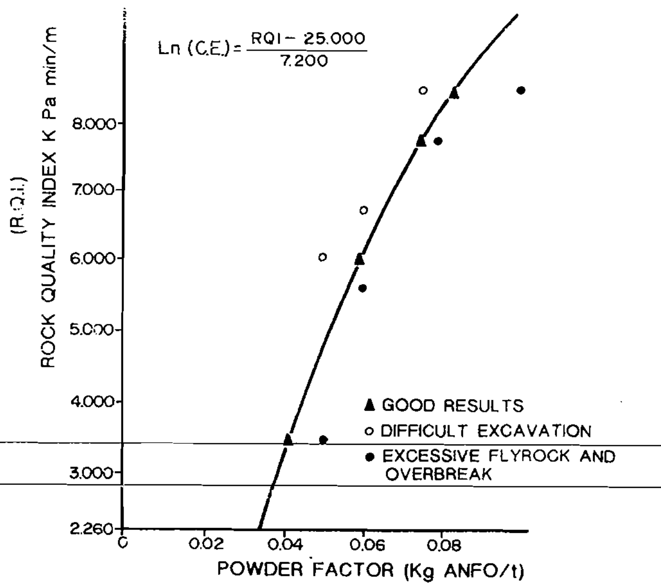


Fig. 18.16. Correlation between Rock Quality Index and Powder Factor for controlled perimeter blasting (Leighton).

The first practical application of the RQI was developed by Little (1975), in an attempt to correlate the rotary drilling data with the geotechnical design of the remaining walls in open pits.

The investigation carried out demonstrated this was not very reliable in view of the recording techniques and in the lack of sensitivity to immediate lithological changes.

Leighton (1982) proceeded to identify the rocks in the Afton mine (Canada) by the RQI, using a B.E. 40-R rotary drill working with a 229 mm (9″) diameter.

Following this he made a study of the correlation between the RQI and the optimum powder factor for perimeter blastings, obtaining a correlation coefficient of $r = 0.98$. Fig. 18.16, for the following exact curve:

$$\mathrm{Ln(CE)} = \frac{\mathrm{RQI} - 25.000}{7200}$$

where: CE = Powder factor (kilograms of ANFO/ton), RQI = Rock Quality Index.

However, the use of the RQI has the following limitations:

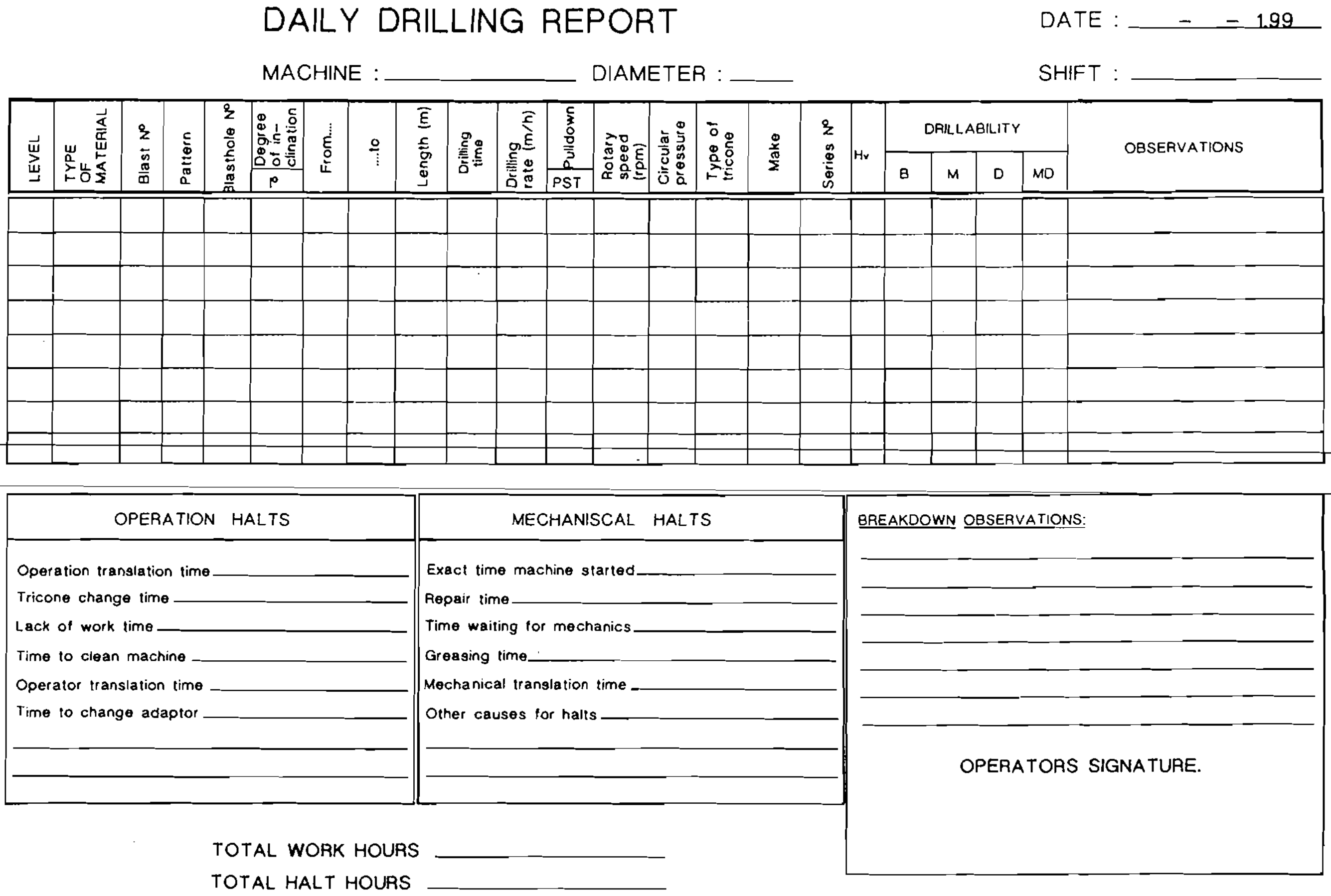
- As the hydraulic pressure of the drill is used, the data obtained depend upon the type and model of the rig.
- The drilling diameter is not taken into account.
- The rotary speed is not a factor.

In conclusion, the results obtained from the Afton mine are only of use in those operations where:

- The drilling rig is model B.E. 40-R, and
- The blastholes are drilled with a diameter of 229 mm.

18.8.3 *Ip drilling index*

E. L. Jimeno (1984), taking into account the limitations of



DAILY DRILLING REPORT

DATE : ___ – ___ – 199_

MACHINE : ____________ DIAMETER : _____

SHIFT : ____________

LEVEL	TYPE OF MATERIAL	Blast Nº	Pattern	Blasthole Nº	Degree of in-clination	From....	....to	Length (m)	Drilling time	Drilling rate (m/h)	Pulldown	Rotary speed (rpm)	Circular pressure	Type of tricone	Make	Series Nº	Hv	DRILLABILITY				OBSERVATIONS
					º						PST							B	M	D	MD	

OPERATION HALTS

Operation translation time ________
Tricone change time ________
Lack of work time ________
Time to clean machine ________
Operator translation time ________
Time to change adaptor ________

MECHANISCAL HALTS

Exact time machine started ________
Repair time ________
Time waiting for mechanics ________
Greasing time ________
Mechanical translation time ________
Other causes for halts ________

BREAKDOWN OBSERVATIONS:

OPERATORS SIGNATURE.

TOTAL WORK HOURS ________

TOTAL HALT HOURS ________

Fig. 18.17. Drilling sheet.

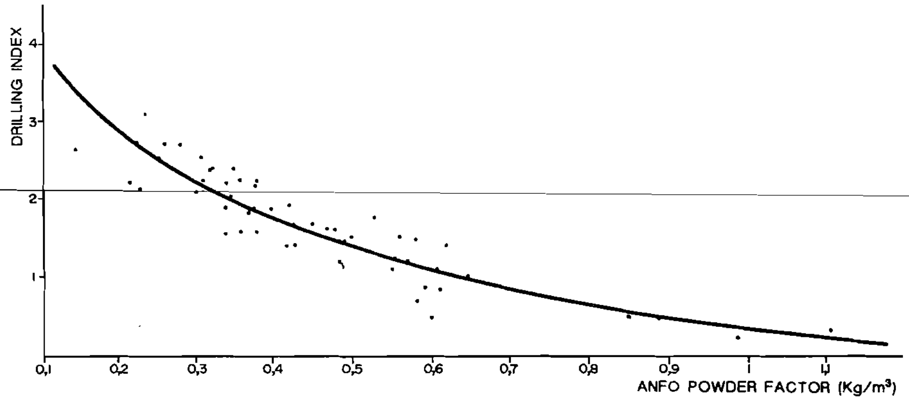


Fig. 18.18. Correlation between the Ip index and the powder factor (L. Jimeno)

the RQI, proposed a rock characterization index in which the following drilling parameters are combined: VP = Penetration rate (m/h), E = Pulldown weight on the tricone bit (thousands of pounds), N_r = Rotary speed (rpm), D = Drilling diameter (inches).

The index responds to the equation:

$$\mathrm{Ip} = \frac{VP}{\dfrac{E \times \mathrm{N}_r}{D^2}}$$

In order to calculate this index, it must be taken into consideration that:

– The type of tricone bit used is the most adequate for the rock formation that is going to be drilled.

– There is sufficient flow of air to sweep away the drill cuttings.

– The following times not be counted: changing drill position, changing the rods, etc. Only the net penetration rate should be taken.

To compile all the data, a sheet like the one shown in Fig. 18.17, is very useful.

As the penetration rate depends upon the compressive, tensile and shearing strengths, the Ip index, which is directly proportional to VP, will implicitly contain all these geomechanic characteristics, which can then be correlated with the specific charge or energy factor used in blastings where adequate fragmentation has been obtained. Fig. 18.18.

The statistic analysis of regression of the data from

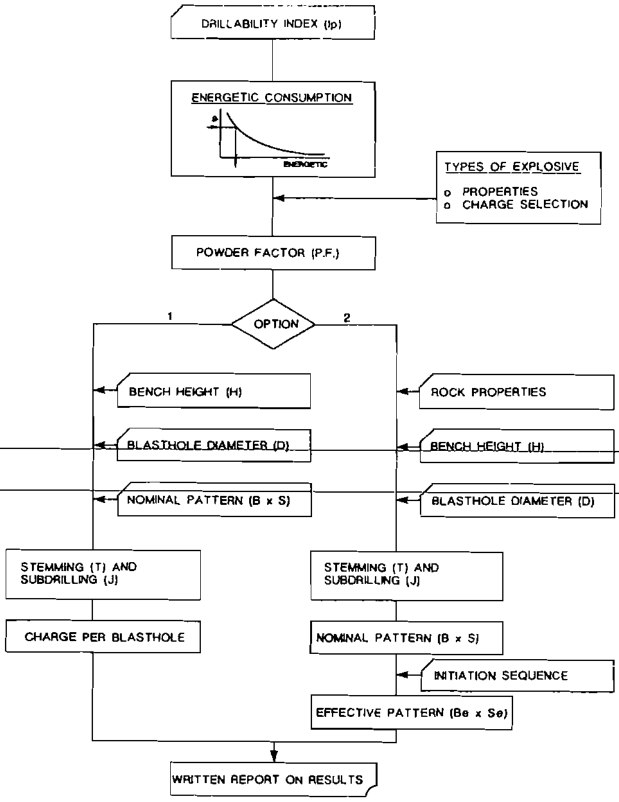


Fig. 18.19. Calculation of the drilling and charging patterns from Ip (L. Jimeno).

Disvol Program Pattern Calculation.

Data			
Drilling diameter	=	311.0 mm	
Bench height	=	15,0 m	
Drilling index	=	0.35	
Explosives	=	(2)	(1)
Charge density (G/CC)	=	0.80	1.35
Detonation velocity	=	4000	4700
Charge diameter	=	311.0	311.0
Results			
Drilling length	=	17.48 (m)	
Upper stemming height	=	7.77 (m)	
Intermediate stemming height	=	0.00 (m)	
Subdrilling length	=	2.48 (m)	
Bottom charge length E-1	=	2.48 (m)	
Bottom charge length E-2	=	7.22 (m)	
Upper charge length E-2	=	0.00 (m)	
Bottom charge E-1	=	255.15 (kg)	
Bottom charge E-2	=	439.07 (kg)	
Upper charge E-2	=	0.00 (kg)	
Total charge	=	694.22 (kg)	
Burden	=	6.99 (m)	
Spacing	=	8.04 (m)	
Volume per blasthole	=	844.02 (m^3)	
Drilling yield	=	86.89 (m^3/m)	
Powder factor (ANFO)	=	0.91 (kg/m^3)	

Fig. 18.20. Calculation of the blasting pattern from the Ip.

Disvol Program Charge Calculation.

Data		
Drilling diameter	=	229.00 mm
Bench height	=	15.00 mm
Drilling index	=	3.00
Burden	=	8.50 m
Spacing	=	9.50 m
Subddrilling	=	1.80 m
Results		
Drilling length	=	16.80 (m)
Upper stemming length	=	10.30 (m)
Intermediate stemming length	=	0.00 (m)
Bottom charge length E-1	=	1.07 (m)
Bottom charge length E-2	=	5.41 (m)
Upper charge length E-2	=	0.00 (m)
Bottom charge E-1	=	53.16 (kg)
Bottom Charge E-2	=	178.56 (kg)
Upper charge E-2	=	0.00 (kg)
Total charge	=	231.72 (kg)
Volume per blasthole	=	1211.25 (m^3)
Drilling yield	=	72.09 (m^3/m)
Powder factor (ANFO)	=	0.20 (kg/m^3)

Fig. 18.21. Calculation of the explosive charges for a prefixed pattern.

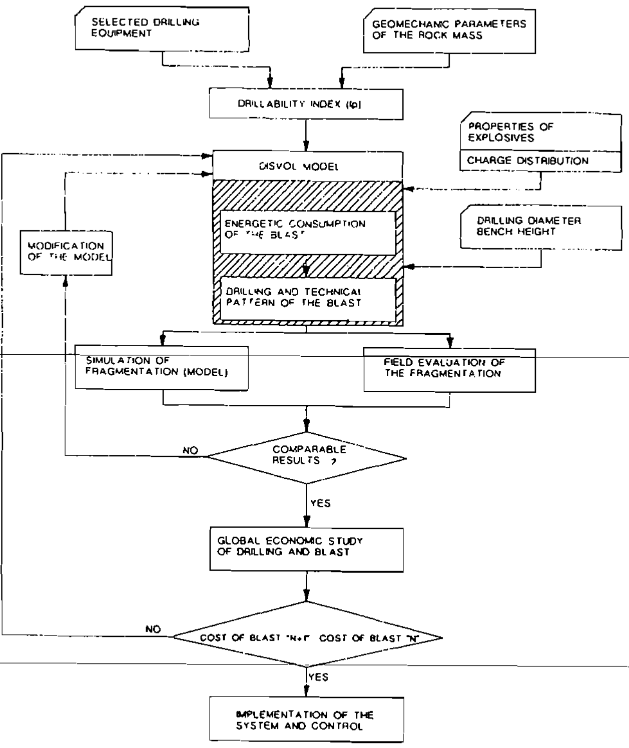

SELECTED DRILLING EQUIPMENT
GEOMECHANIC PARAMETERS OF THE ROCK MASS
DRILLABILITY INDEX (Ip)
PROPERTIES OF EXPLOSIVES
CHARGE DISTRIBUTION
DISVOL MODEL
ENERGETIC CONSUMPTION OF THE BLAST
DRILLING DIAMETER BENCH HEIGHT
MODIFICATION OF THE MODEL
DRILLING AND TECHNICAL PATTERN OF THE BLAST
SIMULATION OF FRAGMENTATION (MODEL)
FIELD EVALUATION OF THE FRAGMENTATION
NO
COMPARABLE RESULTS ?
YES
GLOBAL ECONOMIC STUDY OF DRILLING AND BLAST
NO
COST OF BLAST "N+1" COST OF BLAST "N"
YES
IMPLEMENTATION OF THE SYSTEM AND CONTROL

Fig. 18.22. Structure of the cost optimization model (L. Jimeno).

numerous mines has given the following equation:

$$\text{CE (kg ANFO/m}^3) = 1.124 \times e^{-0.5727\,\text{Ip}} (r = 0.92)$$

The range of rocks checked goes from the very soft, such as the surface of Puertollano, up to the very hard, such as the cupriferous porphyry of Palabora. It must be pointed out that the fundamental taking of data was carried out in the Meirama operation with shales and granites which varied greatly in degrees of weathering. The previous equation which links the powder factor with the drilling index Ip, constitutes a very efficient tool for use in design and calculation of the blastings, as it permits:

- The selection of the drilling pattern, Figs. 18.19 and 18.20.

- Calculation of the optimum charge of a blasthole drilled according to an established pattern, Figs. 18.19 and 18.21, and

– The creating of an optimization model for bench blasting.

Other possibilities of the Ip index are:

– The geotechnical characterization of the open pit materials.

– Designing of the open pit walls.

– The selection of the drilling diameter and the drill characteristics as a function of: The required production, rock strength, and the calculation of grinding yield and costs.

The advantages that the use of the drilling index Ip brings to a model of blasting calculation are the following:

a) As a method for designing the blasting pattern it includes the following data:

Geometric

- Bench height
- Drilling diameter
- Subdrilling, fixed as a function of the diameter.
- Stemming, as a function of Ip.

Properties of the rock and of the rock mass

This is the most important point and the one that makes it different as a method from the rest of the classic formulas in existance which only take into consideration certain properties of the rock.

Explosive

The selection of the specific charge with reference to a base explosive such as ANFO, gives a better application and use of these substances, which can be expressed in Kg/m^3 or in cal/m^3.

The planning of a blast pattern from drill performance data can be done:

– By means of the values given by the manufacturers of the tricone bits from representative samples taken during its design stage, or by widening to an area without previous data.

– By means of interpolation with data compiled from a working operation.

b) It is the only calculation system that determines the explosive charge per blasthole when the rock characteristics are different than imagined during drilling. It is a first step in powder factor optimization and selection of the most adequate explosive agents for the work at hand.

c) It is a reliable method, as it is based upon a stadistic analysis of a large number of samples in which a correlation coefficient that is close to the unit has been obtained for the exact curve, taken from diverse rocks and operations.

d) Combined with a system of continuous monitoring of the drilling and a microprocessor, the range of possibilities for its use can be widened, as indicated before.

18.9 SYSTEM OF DRILLING DATA MANAGEMENT IN ACTUAL TIME

Recently, in the coal mine of Puertollano, a system of recording operation data in the actual time of a rotary drill has been perfectioned.

The group of parameters controlled is:

Parameters all/nothing:

- Motor of the drill running YES/NO
- Mast down YES/NO
- Air in blasthole YES/NO
- Pulldown weight in blasthole YES/NO

Analogical parameters:

- Shifting of the machine
- Shifting of the drilling head
- Rotary speed
- Rotary torque
- Pulldown weight

In order to automatically obtain all the abovementioned parameters, sensors have been placed upon the drill as indicated in Fig. 18.23.

The final configuration of the management system is shown in the block diagram of the Fig. 18.24. The central station is composed of a microcomputer which has a color monitor, an expanded keyboard and printer, which also has an interface for communication with the radio telemetric.

The mobile unit upon the machine contains the cap-

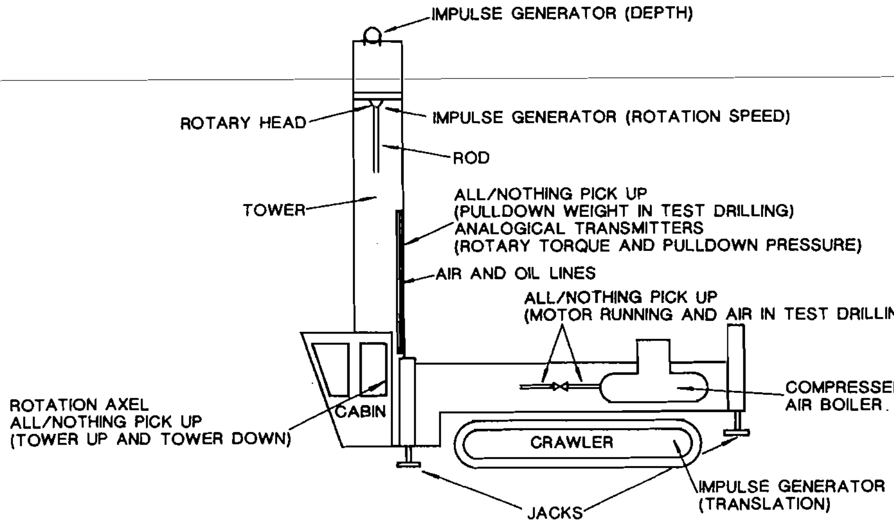


Fig. 18.23. Location of the sensors on the drill.

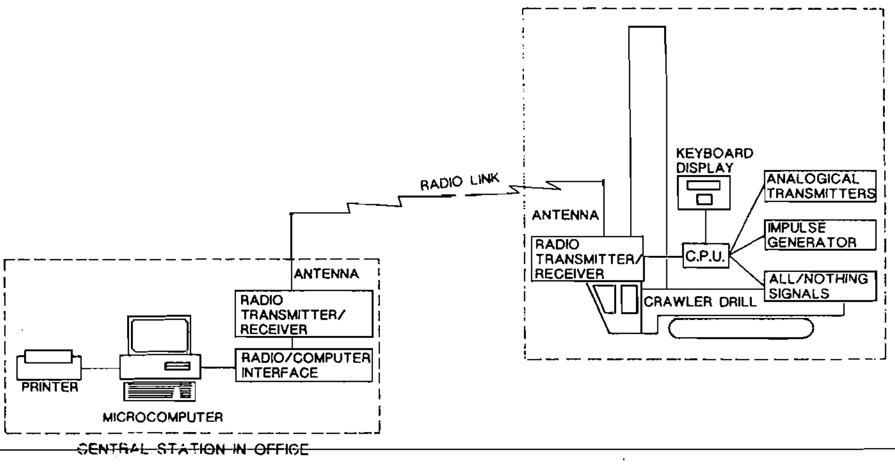


Fig. 18.24. Block diagram of the system of monitoring and controlling drilling.

tators, the CPU and the radio transmitter-receiver. Part of the information obtained is on the display during drilling, to help the operator. The data that appear are:

- The depth of the blasthole at the moment (m)
- Penetration rate (m/s)
- Tricone distance from the bottom of the blasthole.

With the information received in the central station, diverse reports are prepared: lists of work halts, reports of weekly or monthly work shifts, etc. Apart from this, analogical graphics of the blastholes are obtained in which the following parameters are represented:

- Rotary speed.
- Rotary torque.
- Pulldown weight.
- Penetration rate.
- Specific energy of pulldown weight and of rotation.
- Total specific energy.

All the information is stored in the hard disk of the microcomputer, for later use in the blast design, once the drilled rock materials have been characterized.

REFERENCES

Broadbent, C.D.: *Predictable blasting with in-situ seismic surveys.* Mining Engineering, April, 1974.

Borquez, G.V.: *Estimating drilling and blasting – An analysis and prediction model.* EMJ, January, 1981.

Ghose, A.K.: *Design of drilling and blasting subsystems – A rock mass classification approach.* Mine Planning and Equipment Selection, Balkema, 1988.

Garcia-Sineriz, J.L. & M. Colomo: *Sistema de Gestión de Datos en Tiempo Real de la Operación de Máquinas Móviles en Minería a Cielo Abierto.* Canteras y Explosivos, Octubre, 1989.

Hagan, T.N. & R.W. Reid: *Performance monitoring of production efficiency.* Second International Surface Mining And Quarrying Symposium, Bristol, 1983.

Hagan, T.N. & I.M. Gibson: *Using geophysical logs in highwall blast design.* Bulletin of the International Association of Engineering Geology, Paris, 1983.

Leighton, J.C.: *Development of a correlation between rotary drill performance and controlled powder factor.* CIM, Bulletin. August, 1982.

Lilly, P.A.: *An empirical method of assessing rock mass blastability.* Julius Kruttschnitt Mineral Research Center, 1986.

Lopez Jimeno, E.: *Caracterización del Macizo Rocoso en Relación con el Diseño de Voladuras.* Canteras y Explotaciones. Abril, 1985.

Lopez Jimeno, E.: Implantación de un Método de Cálculo y Diseño de Voladuras en Banco. Tesis Doctoral, E.T.S. de Ingenieros de Minas. Madrid, 1986.

Lopez Jimeno, E. & E. Muñiz: *A new method for the design of bench blasting.* Second International Symposium on Rock Fragmentation by Blasting. Colorado, 1987.

Praillet, R.: *A new approach to blasting.* Drilltech Inc.

Pfister, D. & P. Hamelin: *Digital recording of drilling parameters.* Soletanche Enterprise, 1984.

CHAPTER 19

Controllable parameters of blasting

19.1 INTRODUCTION

In calculation and design of blasting, the controllable parameters are classified in the following groups:

A. *Geometric* (Diameter, charge length, burden, spacing, etc.).

B. *Physicochemical* or pertaining to explosives (Types of explosives, strength, energy, priming systems, etc.).

C. *Time* (Delay timing and initiation sequence).

In order to facilitate comprehension and normalize the symbols used in this text, Fig. 19.1 is used to ilustrate a bench blasting where the different parameters of design and the most frequent terminology used in fragmentation with explosives are shown.

Design parameters

H = Height of bench
D = Blasthole diameter
L = Drilled length of blasthole
d = Diameter of explosive charge
B = Nominal burden
S = Nominal spacing
LV = Length of the blast area
AV = Width of the blast area
B_e = Effective burden distance
S_e = Efective blasthole spacing
T = Stemming length
J = Subdrilling
l = Charge length
θ= Angle sustented at the center of the shot by charge fired in the same delay
v/w = Degree of balance of the effective staggered blasthole
t_r = Delay timing
1 = Toe
2 = Half cast
3 = Overhang
4 = Overbreak or backbreak
5 = Tension crack
6 = Cutoffs
7 = Fizzle crater
8 = Decoupled charge

Below, the influence of each of the indicated parameters on the results of blasting is explained, along with the present day trends and why they were chosen.

19.2 BLASTHOLE DIAMETER

The ideal drilling diameter for a given operation depends upon the following factors:

- Properties of the rock mass to be blasted.
- Degree of fragmentation required.
- Height of bench and configuration of charges.
- Costs of drilling and blasting.
- Capacity of the loading equipment.

When the diameter of the holes *D* is small, the costs of drilling, priming and initiation are high and charging, stemming and connection take a lot of time and labor.

When *D* is very small, the only advantage is a lower powder factor due to a more optimum distribution of the explosive.

When the diameters are large, the drilling pattern will be correspondingly large and the size distribution obtained could well be unacceptable if the joints and discontinuities are widely separated and form blocks in situ, Fig. 19.2.

In these cases it is recommended that the spacing between blastholes be smaller than the mean separation between discontinuities. This also applies if the area to be blasted is made up of an elasto-plastic matrix which covers blocks of sound rock which cannot be easily fragmented if they are not intersected by blastholes in a closed pattern.

The increase in *D* is accompanied by the following advantages:

- Higher detonation velocity which gives more stability and is less influenced by external conditions.
- Lower overall costs of drilling and blasting.
- Possible mechanization of the explosive charge.
- Higher drilling productivity (m^3 blastings/ml drilled).
- Increase in shovel yield as a consequence of fewer low productivity zones, Fig. 19.3.

If fragmentation is to remain constant and *D* is increased, it will be necessary to increase the powder factor as the charges are not as well distributed in the rock mass.

The stemming length *T* increases with the drilling diameter, and the collar of the blasthole could become a potential source of boulder formation.

In massive rocks, when the charge length *l* and diameter *D* have ratios of $l/D < 60$, an increase in the latter parameter tends to increase fragmentation. This is explained by the effect of breakage off each end of the short

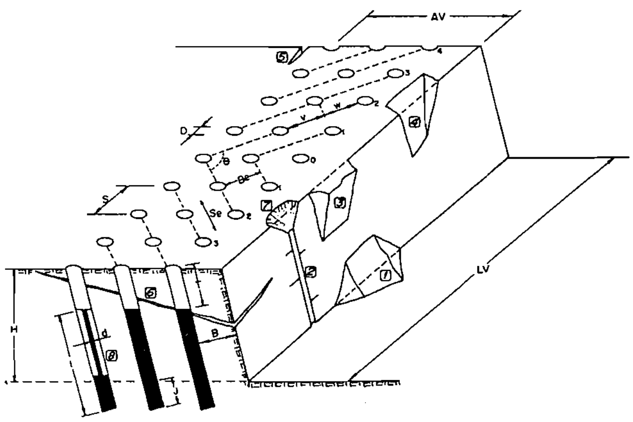


Fig. 19.1. Terminology used in bench blasting.

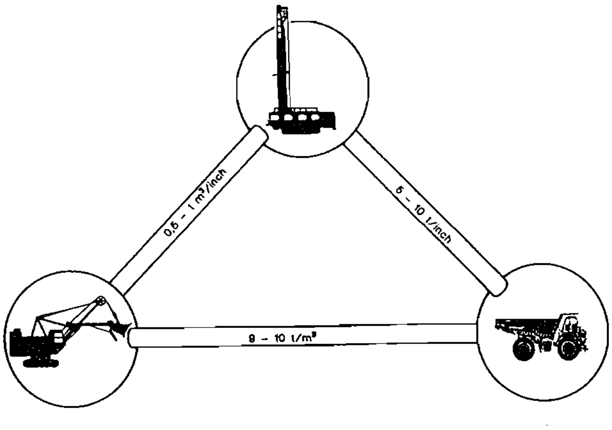


Fig. 19.4. Triangle of dimension balance between drilling equipment, loading and haulage.

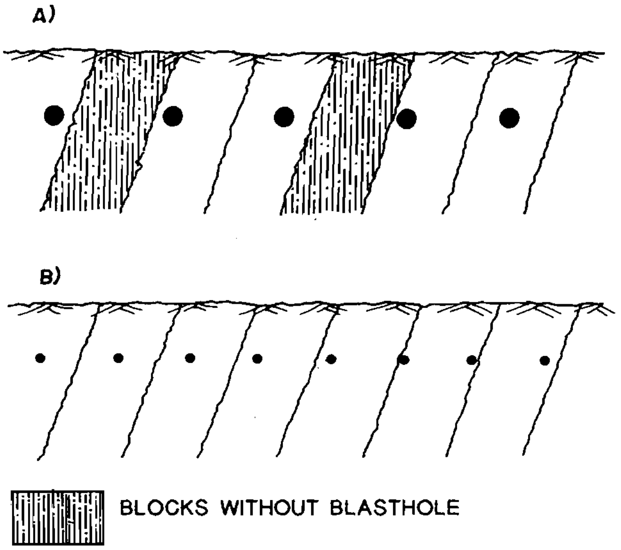


Fig. 19.2. Influence of drilling pattern and discontinuities in the production of large blocks.

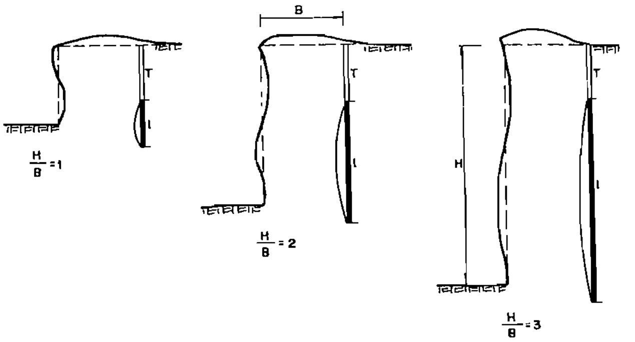


Fig. 19.5. Bending conditions in bench blasting with different *H/B* ratios (Ash).

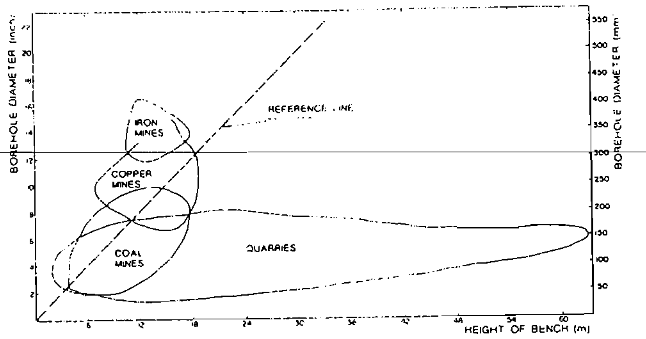


Fig. 19.6. Drilling diameter vs. bench height (Konya and Skidmore, 1981).

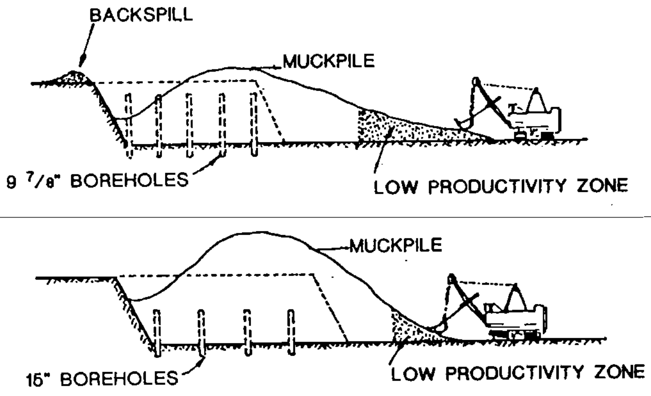


Fig. 19.3. Low productivity zone for shovels.

Photo 19.1. Large diameter blasthole.

cylindrical charges. As l/D increases towards 60, the importance of the hemispherical regions decreases.

When $l/D > 60$, an increase in D requires an increase in the powder factor if fragmentation is to be maintained.

In surface blasts, the diameters cover a wide range which goes from 50 mm up to 380 mm. In civil engineering, it is common to operate with D around 50 to 125 mm, whereas in mining the tendency has been to increase this design parameter and it is not unusual to find diameters between 165 and 310 mm.

In underground operations, the increase in blasthole diameter has been limited and only in metal mining have values between 125 and 220 mm been used. In tunnel and drift driving the range goes from 32 mm to 64 mm and in bench blastings for cave excavations the normal diameter runs from 64 to 90 mm.

When referring to loading equipment, it is necessary to maintain a balance between their dimensions, the drilling diameters and haulage capacity, Fig. 19.4.

19.3 HEIGHT OF BENCH

The stiffness of the parallelepiped of rock located in front of the blastholes exerts great influence on the results of blasting. When the H/B ratio is large, it is easy to displace and deform rock, especially at the bench center. Ash (1977) states that the optimum ratio is $H/B \geq 3$.

If $H/B = 1$, the fragments will be large, with overbreak and toe problems. With $H/B = 2$, these problems are attenuated and are completely eliminated when $H/B \geq 3$.

The condition $H/B \geq 3$ is usually found in quarries and coal strip mining operations, but not in metal mining because the bench height is conditioned by:

- The reach of the loading machine and,
- The dilution of the mineral.

When H is small, any variation in the burden B or spacing S has a great influence on the blasting results. When H increases, with B kept constant, spacing can increase to maximum value without affecting fragmentation.

If the bench heights are very large, there can be problems of blasthole deviation which will not only affect rock fragmentation but will also increase risk of generating strong vibrations, flyrock, and overbreaks because the drilling pattern $B \times S$ will not remain constant in the different levels of the blasthole.

19.4 BLASTHOLE INCLINATION

In bench blasting, inclined drilling gives numerous benefits as well as a few disadvantages which should be studied in each case. Usually, with rotary percussive drilling equipment, the blastholes are inclined; however, in large open pit mines where rotary rigs are used, vertical blastholes seem to be the tendency. The benefits of inclined drilling are:

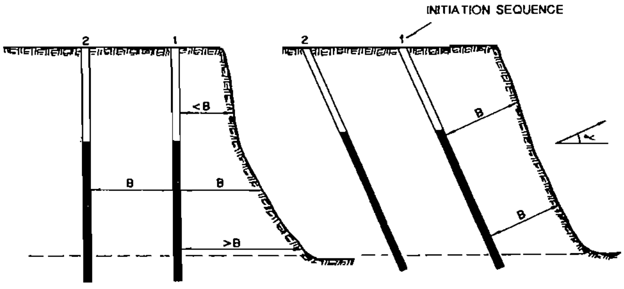


Fig. 19.7. Inclined drilling vs. vertical drilling.

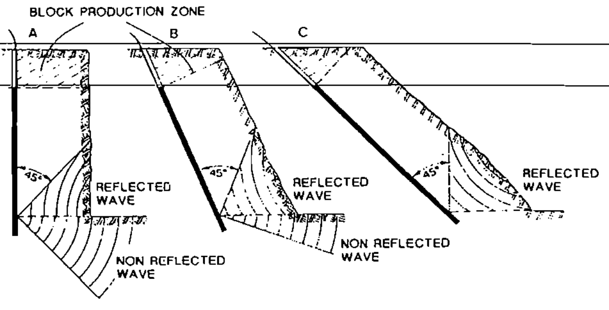


Fig. 19.8. Benefits of inclined holes.

- Better fragmentation, displacement and swelling of the muckpile, as the burden B value is kept more uniform along the length of the blasthole and the angle of the projection direction of the shot increases, Fig. 19.7.
- Less probability of misfire caused by cutoff from burden movement, Fig. 19.7.
- Smoother and sounder slopes in the newly created benches.
- Higher productivity of front end loaders due to more swelling and lower height of the muckpile.
- Less subdrilling and better use of the explosive energy, with the consequent lower vibration level.
- Lower powder factor as the shock wave is reflected more efficiently in the bench toe and the possibility of increasing burden size with less risk of toe appearance, Fig. 19.8.
- In coal mining there is no overcrushing of the mineral when blasting waste.
- Increased drilling production per unit of volume displaced.

On the other hand, the disadvantages are:

- Increased deviation when drilling long blastholes.
- Increased drilling length.
- Difficulty in positioning of the drills and in collaring operations.
- Necessity of close supervision which creates work lapses.
- Lower drill feed which means that in hard rock the penetration rate is limited in direct proportion to the angle of inclination of the mast.
- More wear on the bits, drill steel and stabilizers.
- Less mechanical availability of the drilling rig due to mast fatigue and wear on the traslation system.
- Less productivity with rope shovels due to lower height of the muckpile.
- Poorer flushing of drill cuttings due to friction

forces, requiring an increase in airflow.

– Problems in charging the explosive, especially in blastholes with water.

19.5 STEMMING LENGTH

Stemming is the portion of blasthole which has been packed with inert material above the charge so as to confine and retain the gases produced by the explosion, thus improving the fragmentation process. If stemming is insufficient, there will be a premature escape of the gases into the atmosphere which will produce airblast and danger of flyrock. On the other hand, if the stemming is excessive, there will be a large quantity of boulders coming from the top part of the bench, poor swelling of the muckpile and an elevated vibration level.

In order to determine stemming, the following must be taken into consideration:

– The type and size of the material to be used, and

– The length of the stemming column.

It is common practice to use drill cuttings, owing to their availabitity near the collar of the blasthole. However, recent studies have shown that coarse angular material such as crushed rock is more effective and the resistance to ejection of the stemming column increases when the humidity content is lowered.

The most effective stemming is achieved with particle sizes that range between 1/17 *D* and 1/25 *D*.

Investigations carried out by Otuonye indicate that by using stemming material with a diameter of 1/25 *D* proceeding from crushing, the stemming length can be reduced by up to 41%.

In practice, the optimum lengths of stemming increase as the quality and competence of the rock decrease, varying between 20 D and 60 D. Whenever possible, a stemming length of more than 25 D should be maintained in order to avoid problems of airblast, flyrock, cutoffs, and overbreak.

In multiple row blasts, special care should be taken when stemming front row blastholes, especially when face irregularities are present, as they cause great differences in burden dimensions from top to toe of the bench.

When top priming is carried out, the negative effect of the detonation cord upon the stemming material must be taken into account, as its lateral pressure creates a premature escape route for the venting of gases into the atmosphere.

In underground blastings, with the long blasthole method, the intermediate stemming between sequenced deck charges must be calculated to avoid simultaneous – sympathetic initiation and desensitivation by precompression, in order to maintain a degree of fragmentation that will not interfere with the charge.

19.6 SUBDRILLING

Subdrilling *J* is the length of the blasthole underneath the floor level which is needed to break the rock at bench height and achieve adequate fragmentation and displacement which will allow the loading equipment to reach the expected level of productivity.

If the subdrilling is small, the rock will not be completely sheared off at floor level, which will result in toe appearance and a considerable increase in loading costs.

However, if the subdrilling is excessive, the following will occur:

– An increase in drilling and blasting costs.

– An increase in vibration level.

– Excessive fragmentation in the top part of the underlying bench, causing drilling problems of the same and affecting slope stability in the end zones of the open pit.

– Increase in risk of cutoffs and overbreak, as the vertical component of rock displacement is accentuated.

The breakage in the bottom of the blasthole is produced in the shape of inverted cones, whose angles to the horizontal depend upon the structure of the rock mass and on the residual stresses. Normally they vary between 10 and 30 degrees, Fig. 19.9.

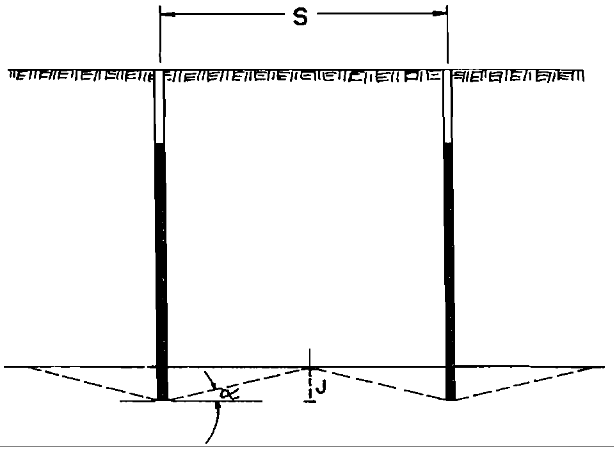


Fig. 19.9. Minimum necessary subdrilling.

Table 19.1.

Different rock formations	J/B
Open bedding plane at toe	
Horizontal stratification	0
Easy toe. Soft rock	0.1-0.2
Normal toe. Medium hard rock	0.3
Difficult toe. Hard rock	0.4-0.5

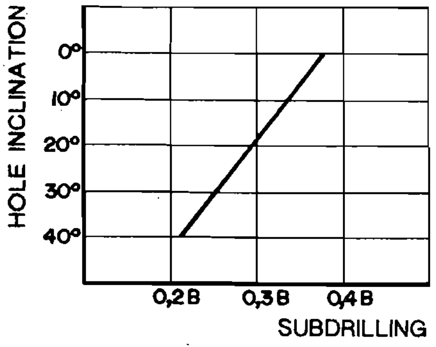


Fig. 19.10. Reducing subdrilling with inclined blastholes.

The value of subdrilling that produces the intersection of the cone shaped surfaces at bench level is usually around $J = 0.3\,B$ because it has been shown that:

$$S = 1 \text{ to } 1.4\,B \text{ and also}$$

$$J = \text{tag } \alpha \times \left(\frac{S}{2}\right)$$

with χ taking on the indicated values.

The normal ratios of J/B for bench blastings are shown in Table 19.1.

In order to reduce subdrilling, the use of explosives which give a high concentration of energy per unit of length in the bottom part of the charge and the drilling inclined blastholes is recommended, Fig. 19.10.

In horizontal bedding plane coal mining operations, in order to eliminate the crushing effect of the ends of the charges, subdrilling takes on a negative value as the bottom of the blasthole is backfilled to a length of approximately 4 *D*.

19.7 BURDEN AND SPACING

The burden *B* is the minimum distance from the axis of a blasthole to the free face, and spacing *S* is the distance between blastholes in the same row. These parameters depend basically upon the drilling diameter, the properties of the rocks and of the explosives, the height of the bench and the desired degree of fragmentation and displacement.

Numerous formulas have been suggested to calculate the burden, which take into account one or more of the indicated parameters; however, their values all fall in the range of 25 to 40 *D*, depending fundamentally upon the properties of the rock mass, Fig. 19.11.

It is very important to be certain that the dimension of the burden *B* is the adequate. Smaller or larger values than those established could appear in the following situations:

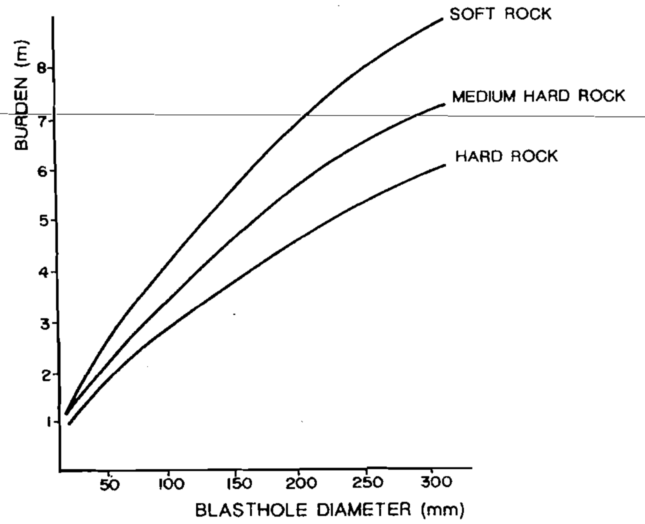


Fig. 19.11. Size of burden in function with the drilling diameter

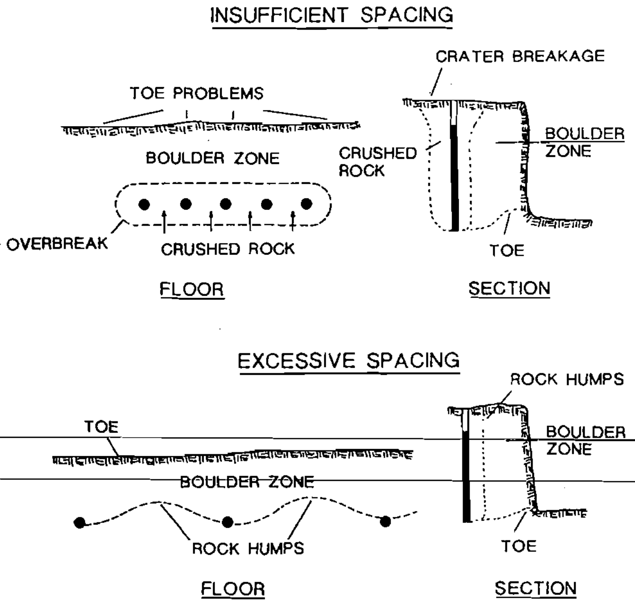


Fig. 19.12. Influence of spacing in bench blasting (Dick and others).

- Marking and collaring errors.
- Inclination and directional errors.
- Deflection errors during drilling.
- Irregularities in the face of the slope.

If the burden is excessive, the explosion gases find too much resistance to effectively fracture and displace the rock and part of the energy becomes seismic and intensifies the vibrations. This phenomenon is most evident in presplitting blasts where there is total confinement and vibration levels can be up to five times those of bench blasting.

If the burden is small, the gases escape and expand at high speed towards the free face, pushing the fragmented rock and projecting it uncontrollably, provoking an increase in overpressure of the air and noise.

Spacing *S* value is calculated in function with the burden, the delay timing between blastholes and the initiation sequence. Very small spacings cause excessive crushing between charges and superficial crater breakage, large blocks in front of the blastholes and toe problems, Fig. 19.12.

Excessive spacing between blastholes causes inadequate fracturing between charges, along with toe problems and an irregular face with overhang in the new bench.

19.8 BLASTHOLE PATTERNS

In bench blastings, the normal blasthole patterns are either square or rectangular, owing to the ease with which the collaring points can be marked out. However, the most effective are *staggered* patterns, especially those drilled on an equilateral triangular grid, as they give optimum distribution of the explosive energy in the rock and allow more flexibility when designing the initiation sequence and the break direction. This pattern produces the best fragmentation, with a spacing ratio of $S = 1.15\,B$

for vertical blastholes and $S = 1.15\ B \times \cos U$, where U is the angle with respect to the vertical in inclined holes.

In a square pattern of 4.5 m per side, Fig. 19.13 (a), the farthest point, equidistant from the four blastholes, is at 3.18 m. In a staggerd equilateral triangular pattern, Fig. 19.13 (b), the equivalent pattern is 4.2 × 4.8 m and the center of the triangle is at a distance of 2.79 m from the blastholes.

As the fall of the stresses produced by the shock wave is proportional to the square of the distance, in the point that is equidistant from the blastholes, with a square pattern, there will be 23% less energy than in an equivalent staggered pattern.

In soft rock, good results are achieved with rectangular patterns and it is usually not necessary to use staggered blasthole patterns.

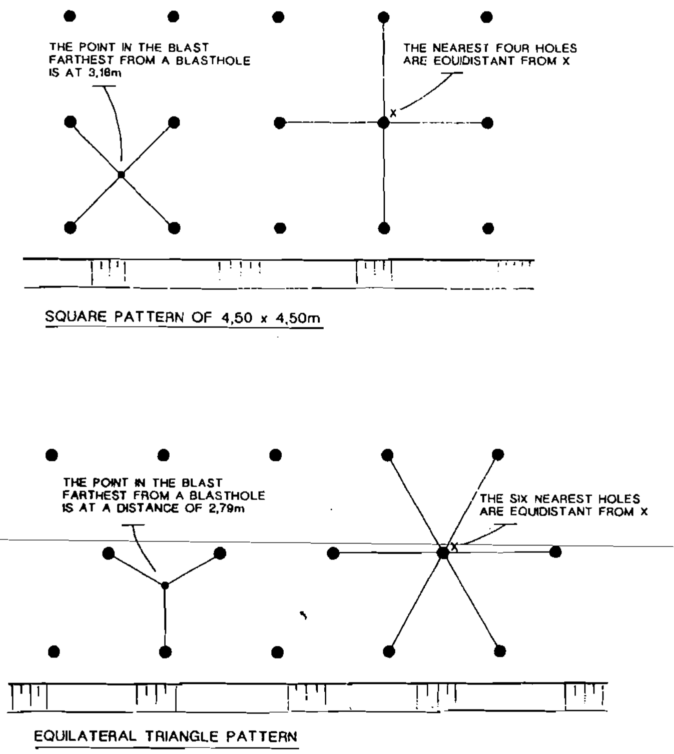

THE POINT IN THE BLAST FARTHEST FROM A BLASTHOLE IS AT 3,18m
THE NEAREST FOUR HOLES ARE EQUIDISTANT FROM X
X
SQUARE PATTERN OF 4,50 x 4,50m
THE POINT IN THE BLAST FARTHEST FROM A BLASTHOLE IS AT A DISTANCE OF 2,79m
THE SIX NEAREST HOLES ARE EQUIDISTANT FROM X
X
EQUILATERAL TRIANGLE PATTERN

Fig. 19.13. Comparison between a square pattern (a) versus a staggerd pattern with equilateral triangles (b).

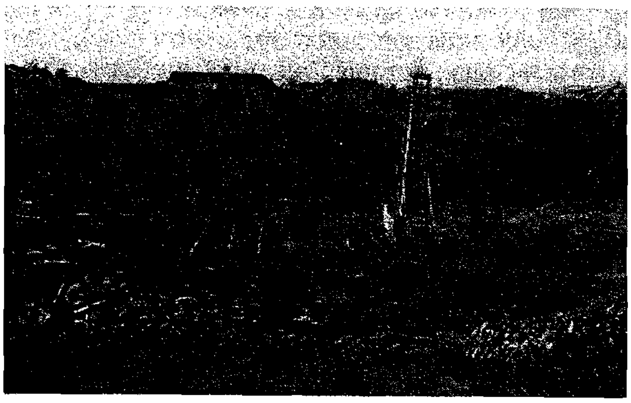


Photo 19.2. Drilling of a large diameter blast.

19.9 GEOMETRY OF THE FREE FACE

The most effective geometry of the free face is that in which each point of its surface is equidistant from the center of the explosive charge. With small spherical charges, this situation exists in secondary blasting where, with a small charge confined in a blasthole, breakage is achieved with a very low powder factor, around 80 to $100 g/m^3$. However, in production blastings with cylindrical charges, the conditions are different and it is necessary to use larger quantities of explosive.

The most effective face geometry is achieved by:

– Placing the charges parallel to the free face or by forming the smallest possible angle.

– Having the rock near the surface already cracked by previous blastings.

– Designing an initiation sequence in which each blasthole has a face that forms a semi-cylindrical convex or bi-planar surface, Fig. 19.14.

In underground blastings, in tunnel driving or raise boring by shooting blastholes against a concave free face, fragmentation and displacement are achieved with great difficulty. This explains why the distances from the lead blastholes to said voids are smaller than would be needed if the free face were planar or bi-planar convex, Fig. 19.15.

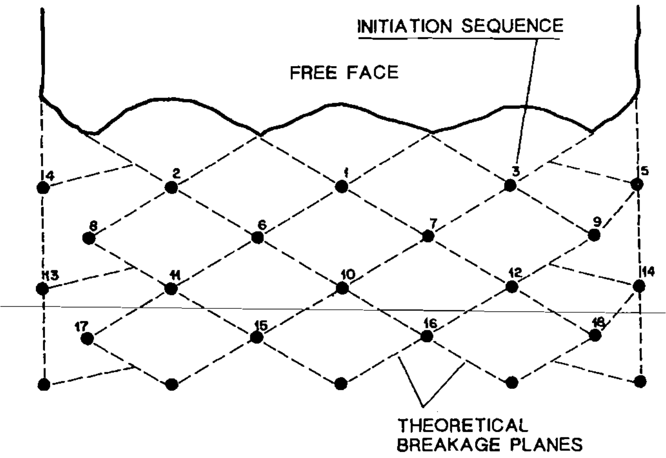


Fig. 19.14. Geometry of the free face and initiation sequence (Hagan).

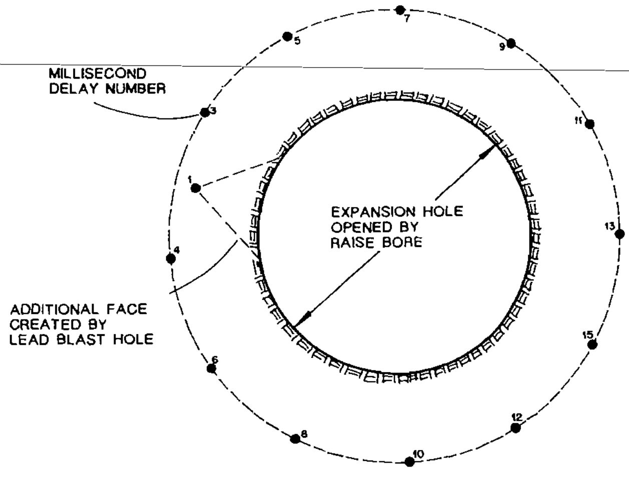

MILLISECOND DELAY NUMBER
1
3
4
5
6
7
8
9
10
11
12
13
15
EXPANSION HOLE OPENED BY RAISE BORE
ADDITIONAL FACE CREATED BY LEAD BLAST HOLE

Fig. 19.15. Raise boring (Hagan).

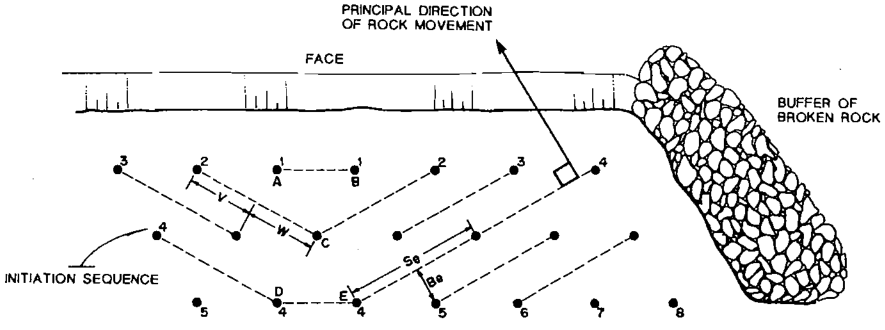


Fig. 19.16. Firing blasts with a face covered by a buffer of broken rock (Hagan).

All necessary auxiliary operations to provide clean faces, without toe, should be carried out before blasting. When a round is fired without having fully cleared away the buffer of broken rock, resting on one of the free faces from the previous blast, the following problems can appear:

– More intense vibrations are generated, more over break and danger of instability.

– To achieve the same degree of fragmentation and swell as with a clean face, a more closed pattern will be necessary and the powder factor will increase.

– Probable toe appearance which will increase loading costs and require secondary blastings.

To overcome some of these inconveniences, the blast should begin in an area that is far away from the covered face and the initiation sequence should be designed with a break direction that is parallel to the face, Fig. 19.16.

19.10 SIZE AND SHAPE OF THE BLAST

The size of the blasts should be as large as possible because of the following advantages:

– Reduction of unproductive times of the drilling and charging equipment, marking out operations, etc.

– Fewer boundries between blasts where there is poorer fragmentation, due to the difficulty in drilling regular blasthole patterns, to the risk of finding preformed blocks from earlier rounds, and to the premature escape of the gases through the existing cracks, apart from the necessary supervision time and control that must take place in any blasting.

On the whole, better fragmentation is usually attained in multi-row blasts than in those of only one row. In underground metal mines, larger blasts also provide better overall results.

The shape of the blastings should be in such a manner that:

– With one free face the ratio length of face/width LV/AV should be larger than 3, Fig. 19.17.

– With two free faces, the blastings should be designed with $LV/AV > 2$, Fig. 19.18.

On the other hand, the problems of multi-row blastings are:

– Increase in intensity of generated vibrations and airblast which, in certain instances, such as proximity to urban areas, are not applicable.

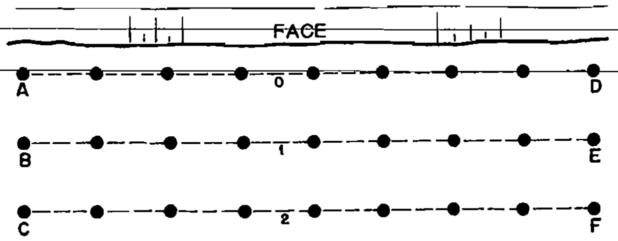


Fig. 19.17. Blast with one free face.

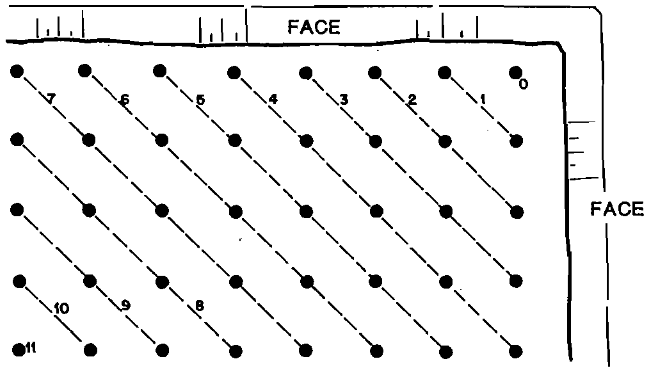


Fig. 19.18. Blast with two free faces.

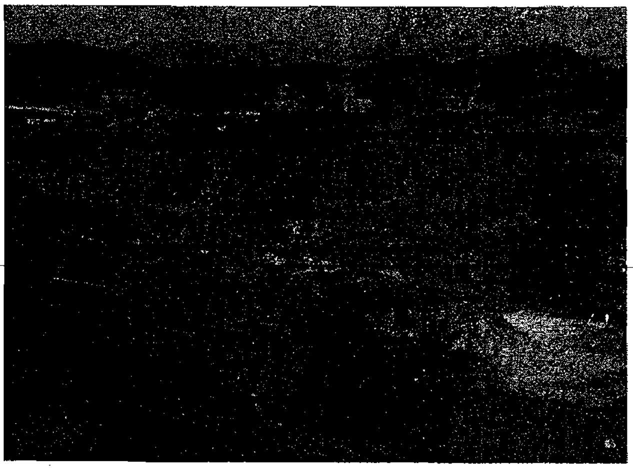


Photo 19.3. Blast with two free faces.

– Appearance of overbreak and fly rock in the back rows if fired with an incorrect sequence.

– In very soft rock, reduction of the benefit of over break volume, as the number of blastings are reduced.

19.11 AVAILABLE EXPANSION VOLUME

When rock is fragmented, there is an increase in volume. If the void where the material expands is less than 15% of the volume, the breakage mechanisms will be adversely affected and the rock fragments will tend to interlace, which results in compressed material.

In underground blasts of great size, it is recommended that the available expansion volume be larger than 25% to achieve an adequate flow of rock towards the loading points and avoid the formation of hung-ups.

In tunnel and drift driving, if the volume of the void in the cut is too small, the phenomenon of sinterization or plastic deformation of the finely broken material will be produced. Whenever possible, it is recommended that the available expansion volume be more than 15% of the actual volume of the cut. In the blasting designs where relief holes are not available, the push of the rock will be obtained by increasing the charge factor in that zone.

19.12 CHARGE CONFIGURATION

When blastholes are short, continuous charges will be used; but if they are quite deep, the best cost/effectiveness relationship is obtained with deck charges.

Harries and Hagan (1979) have demonstrated that the strain generated by detonation of an explosive charge increases when the ratio l/D is enlarged from 0 to 20, remaining constant from there on. This way, using the ratio $l/D = 20$, maximum fragmentation is obtained along with optimum burden value B_o. If these charges are initiated in the central points, an intense fragmentation will be achieved in the hemispherical zone off each of their ends, concluding that a continuous charge with the ratio $l/D = 52$, is no better than a deck charge with a ratio $l/D = 20$, and a stemming of 12 D, Fig. 19.19.

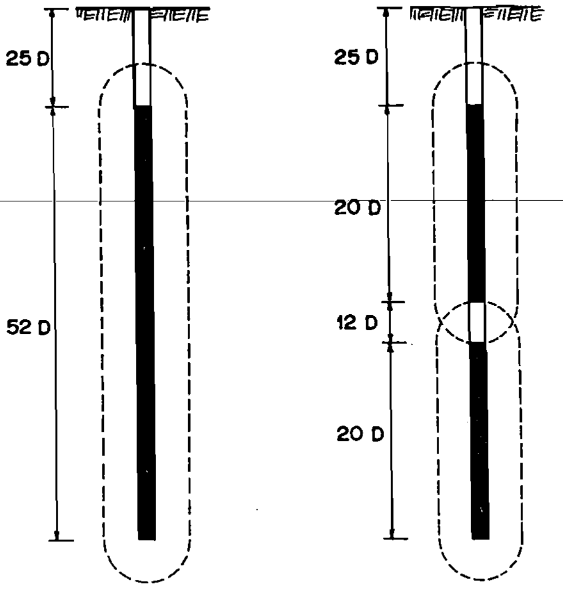


Fig. 19.19. Continuous and decked charges.

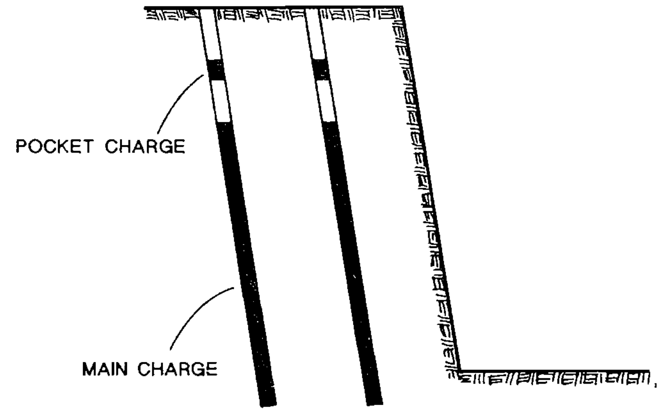


Fig. 19.20. Short pocket charges in the stemming end of the blast.

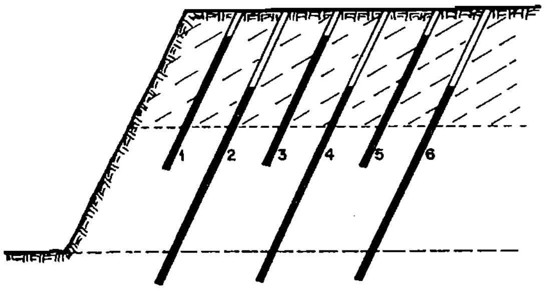


Fig. 19.21 Improving bench top fragmention with additional smaller holes in drilling pattern.

However, the use of deck charges can adversely affect performance of front end loaders as a consequence of reduced displacement and swelling of the material. Where this problem is not a decisive factor, then the decision to use this type of charge configuration will depend upon the difference between the potential savings in explosives and time, the degree of complexity, and initiation costs involved in deck charging.

The appeal of deck charging will increase as explosives become more expensive, stemming becomes mechanized and when vibrations are a conditioning factor, making it necessary to reduce the charge weight per hole by decked and sequenced explosives in the same blasthole.

In surface operations, the minimum bench height necessary for effective decked charges should be such as to give the ratio $H/D > 70$.

In blastings where large blocks result from stemming, short pocket charges should be used, Fig. 19.20.

When vertical drilling is carried out and the rock horizon in the zone of stemming is of greater strength than in the rest of the bench, it might be useful to drill relief or auxiliary blastholes which improve fragmentation, Fig. 19.21.

19.13 DECOUPLING OF THE CHARGES

The Pressure-Time profile of the explosion gases can be controlled for the use of cartridge explosives with two

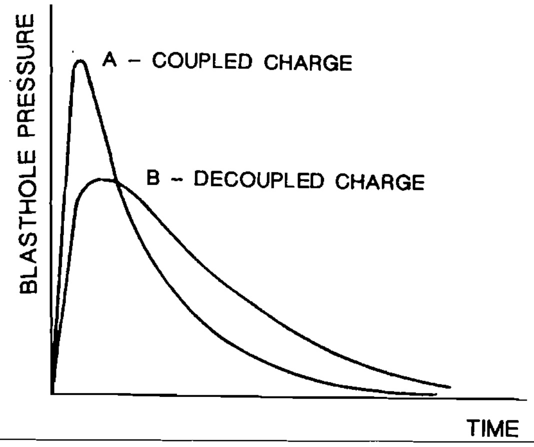


Fig. 19.22. Blasthole Pressure-Time profile with or without charge decoupling.

techniques known as 'Decoupling and Spacing the charges'. The first consists in leaving an empty space, or with inert material, between the explosive column and the wall of the blasthole. The second is based upon dividing the charge by means of air charge decks or dry, granular material.

The experiments carried out by Melnikov (1972), using 65 to 75% air decking, show that in some types of rock there is a marked improvement in fragmentation and uniformity of size distribution, lowering the percentage of secondary blastings from two to ten times, as well as the powder factor and vibration intensity.

The effective pressure of the gases on the blasthole wall with decoupled or decked charges is given by:

$$PB_e = PB \times (V_e/V_b)^{1.2}$$

where: P = Blasthole pressure, V_e and V_b = Volume of the explosive and of the blasthole, respectively.

At the present time, the trends for controlling the pressure profile consist of using bulk explosives to which a diluting substance is added in the appropriate proportion. This method takes less time, is more cost-effective and is being widely used in contour blastings.

19.14 EXPLOSIVES

When choosing an explosive for a given operation, careful attention must be paid to the properties of the rocks to be fragmented, as well as to the explosives available on the market.

When massive rocks are broken, almost all of the specific area of the material is created by the blast, and the explosives used should be of maximum strength and detonation velocity VD which produce high blasthole pressure. On the other hand, in highly stratified or fissured rocks, in which the total area of discontinuities is relatively larger than that created by the blasting, explosives of low density and detonation velocity are more efficient.

Given that, apart from fragmentation, there must be swelling and displacement of the material in order to get good loading productivity, each case must be studied to determine the balance between strain energy ET and bubble energy EB. For a given explosive, these energies will depend upon the diameter of the charges, their density and initiation system.

Emulsions or slurries have a high ET value and should be applied in hard massive rocks and in those situations where displacement is not an important factor, such as in VCR blasting, where the pieces of rock fall from the effects of gravity.

When the detonation of an explosive creates too many fines, due to efect of crushing of the rock, low density agents, such as ANFO and the mixtures of ANFO with inert substances should be used.

19.15 DISTRIBUTION OF EXPLOSIVES IN THE BLASTHOLES

In bench blasting, Fig. 19.23, the necessary energy needed to produce rock breakage is not constant through-

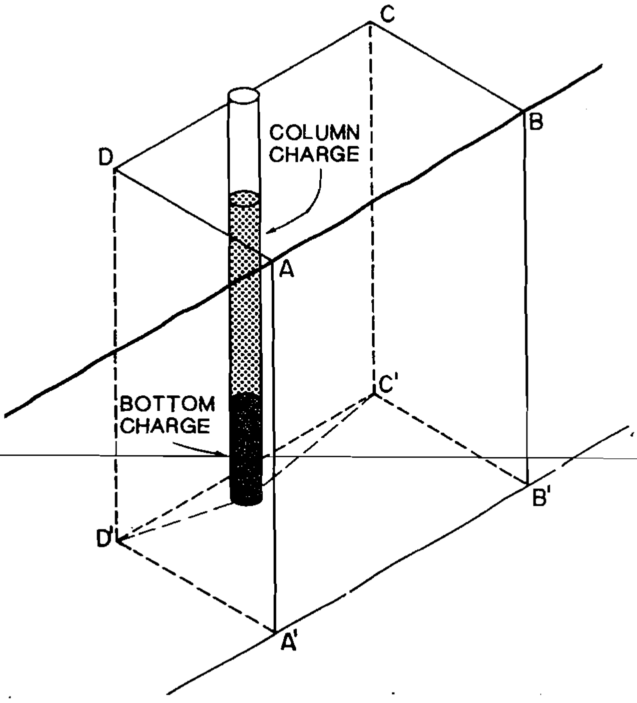


Fig. 19.23. Distribution of the explosive in a blasthole for rock fragmentation in bench blasting.

Photo 19.4. Preparation of the bottom charge.

out. In effect, the energy generated by the explosive should overcome the tensile strength of the rock in the section C DD'C' and the shear strength in section A'B'C'D'.

As the shear strength is greater than the tensile strength, a selective charge distribution should be used so that the specific energy in the bottom of the blasthole be from 2 to 2.5 times greater than the column energy. This means that explosives of high density and strength should be used in the bottom charges, such as dynamites, watergels (slurries) and emulsions; and medium strength, low density explosives in the column charge, such as ANFO or low density watergels and emulsions.

The bottom charge should have a length of at least 0.6 B, so that its center of gravity is above or at least at the same level as the bench floor. According to Langefors, extension of the bottom charge to a length more than equal to the burden value does not contribute appreciably to breakage at bench toe level, therefore suggesting that the lower charge should be between 0.6 and 1.3 B.

While the use of selective charges has customarily been with diameters under 165 mm, in which Langefors' theory of elongated cyndrical charges is applied, in surface mining, with large diameters of 229 to 415 mm, the use of continous charges of bulk ANFO has been widespread and, in some instances, with selective bottom charges of aluminized ANFO, watergels or emulsions with lengths of 8 to 16 D.

In these cases, the use of selective charges have the following advantages:

- Increased drilling productivity as a consecuence of a larger pattern and a smaller length of subdrilling.
- Better bottom breakage which eliminates toe problems and favors the loading operation.
- Lower drilling and blasting costs, especially in hard rock.
- Lower powder factor due to improved use of the same.

19.16 POWDER FACTOR

The quantity of explosive necessary to fragment 1 m^3 or 1 t of rock is the parameter known as *Powder Factor or Specific Charge CE*. According to the opinion of numerous specialists, this parameter is not the best and only tool used to design the blasts, unless it is referring to a pattern explosive or is expressed as energetic consumption, basically because the spatial distribution of the explosive charges within the rock mass has great influence upon the results of the blasts.

The *CE* of a blasting is increased by:

- An increase in the diameter of the blastholes, the rock strength and degree of fragmentation, displacement and swelling desired.
- A poor charge distribution, lower resistance to the ejection of the stemming, shooting against a concave, biplanar free face, or with a buffer of broken rock, inadequate Length/Width ratio and unsatisfactory effective delay timimg of the charge.

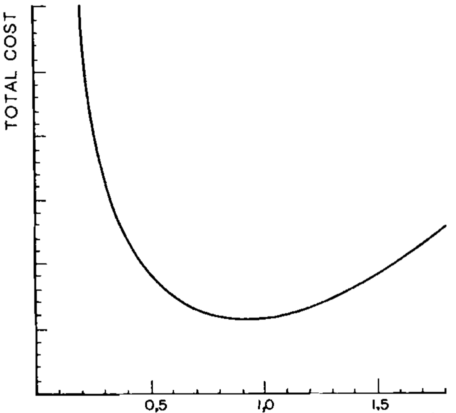


Fig. 19.24. Reduction of operative costs with the powder factor.

Table 19.2.

Types of rock	Powder factor (kg/m^3)
Massive high strength rocks	0.6-1.5
Medium strength rocks	0.30-0.6
Highly fissured rocks, weathered or soft	0.10-0.30

When using blastholes that are parallel to the free face and equilateral triangular patterns, initiated with sequences in V1 and V2, the specific powder factors will be lower.

When *CE* is high, apart from giving good fragmentation, displacement and swelling of the rock, there are less toe problems and it helps to achieve the optimum cost-effectivness of the operation, i.e. drilling, blasting, loading, haulage and crushing, Fig. 19.24.

In Table 19.2, the typical values of *CE*, in different types of rock, for surface bench blasting are given.

In underground blastings, the *CE* can vary between 0.9 and 7 kg/m^3, depending upon the type of rock, free area, diameter of the blasthole and types of cuts.

19.17 INITIATION AND PRIMING

For a given explosive, by use of initiating devices or primers, the balance between the *ET* and the *EB* developed during blasting can be varied, in order to accomodate the characteristic strengths and structures of the rocks.

Also, when an increase in strain energy is required for a section of harder rock within the blasthole, initiating devices can be placed at those levels.

The types of devices, their effects on the charges and their positioning have been discussed in Chapter 14.

19.18 DELAY TIMING AND INITIATION SEQUENCES

The delay times between blastholes and the initiation sequences play an important role in blasting because they reduce the charge weight per hole and, as a consequence, the levels of vibrations produced along with a more effective breakage and control over rock displacement, subdrilling, and flyrock. Later on, these design parameters will be analized in detail.

19.19 INFLUENCE OF LOADING EQUIPMENT ON THE DESIGN OF THE BLASTS

The results of a blast affect the productivity of the different types of loading equipment, not only because of the size distribution of the material, but also because of its swelling and the geometric profile of the muckpile. When rope or hydraulic shovels are used, the height of the bench will be the deciding factor for efficiency of the machines and the blasts should be designed so as to provide adequate fragmentation and a muckpile that is not too extended with few low productivity zones, Fig. 19.25.

- too clean
- low productivity with shovel
- high productivity with front end loader
- very safe
- not very clean
- high productivity with shovel
- low productivity with front end loader
- dangerous
- not very clean
- acceptable productivity
- safe

If front end loaders are used, the tendency will be towards a type of blasting that produces maximum displacement and swelling of the rock, high fragmentation and reduced height of the muckpile, Fig. 19.25.

In surface coal operations or with horizontal bedding planes where draglines are used, the present tendency is to use cast blasting which avoids use of a large boom, lowering costs and reducing the volume of rock requires rehandling.

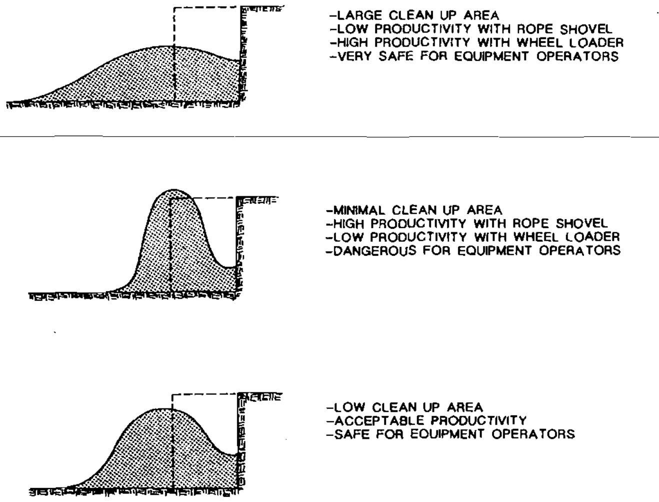

-LARGE CLEAN UP AREA
-LOW PRODUCTIVITY WITH ROPE SHOVEL
-HIGH PRODUCTIVITY WITH WHEEL LOADER
-VERY SAFE FOR EQUIPMENT OPERATORS
-MINIMAL CLEAN UP AREA
-HIGH PRODUCTIVITY WITH ROPE SHOVEL
-LOW PRODUCTIVITY WITH WHEEL LOADER
-DANGEROUS FOR EQUIPMENT OPERATORS
-LOW CLEAN UP AREA
-ACCEPTABLE PRODUCTIVITY
-SAFE FOR EQUIPMENT OPERATORS

Fig. 19.25. Profiles of the muckpile after blasting.

19.20 SPECIFIC DRILLING

Specific drilling is defined as the drilled hole volume or drilled hole length drilled per volume unit of rock. As in other design parameters, specific drilling is in function with the blastability of the rock.

The equation used to calculate the specific drilling *PS* in ml/m^3 is:

$$PS = \frac{(H/\cos\beta + J)}{\dfrac{B}{\cos\beta} \times S \times H}$$

where: H = Bench height (m), J = Subdrilling (m), B = Burden (m), S = Spacing (m), β = Blasthole inclination (degrees), and if *PS* is to be obtained in l/m^3, the following is applied:

$$PS = \frac{(H/\cos\beta + J) \times [250 \times \pi \times D^2]}{\dfrac{B}{\cos\beta} \times S \times H}$$

where: D = Drilling diameter (m). In Fig. 19.26 the Specific Drilling is expressed as function of D and the blastability of the rock.

1. Very easy to blast.
2. Easy to blast.
3. Not so easy to blast.
4. Difficult to blast.

Blasthole diameter (mm).

Fig. 19.27, represents a nomograph from which the volume of blasted rock per drilled lineal meter can be calculated from *PS*, according to the diameter of the blastholes and reciprocally.

Yield of rock per drilled meter hole (m^3/ml).

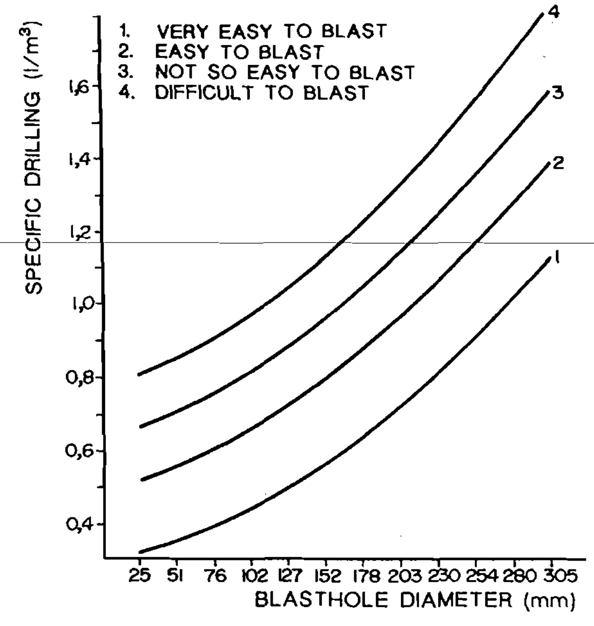


Fig. 19.26. Specific drilling of four types of rock in 12 m high benches.

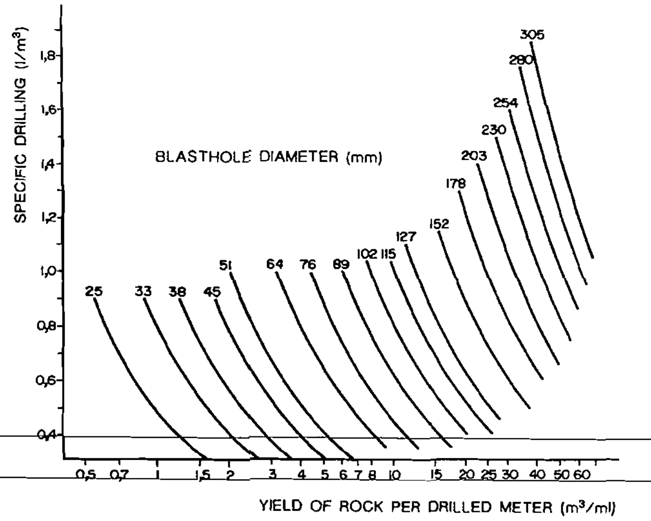


Fig. 19.27. Relationship between yield of blast and specific drilling.

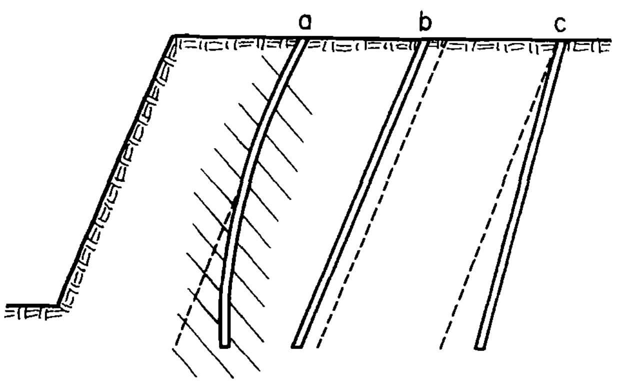


Fig. 19.28. Deviation of blastholes and drilling errors.

Table 19.3.

System of alignment	Alignment error (%)	
	Normal	Careful
Manual	4-7	3-5
With indicator	0.5-1.0	0.2-0.5

Photo 19.5. Stabilizer of blasthole rotary drill.

19.21 BLASTHOLE DEVIATION

The factors which cause blasthole deviation can be classified in four groups, Fig. 19.28.

First of all, the structural properties of the rock, such as the schistosity planes, the fissures, the loose open joints filled with soft materials, the lithological changes, etc. This group is especially important when the drilling direction is oblique to these planes.

Secondly, if the chosen bit diameter is too large in comparison with the diameter of the drill steel, a deviation of the blastholes is produced due to lack of bending resistance in the drill string and a premature wear of the same.

And, last of all, the errors in collaring and alignment, which are the most common in drilling operations.

In collaring errors, it is frequent to find deviations of more than 10 cm or typically about one hole diameter. Alignment errors are usually caused by improper setting of the feeds. Some typical values are shown in Table 19.3.

REFERENCES

Ash, R.L.: *The mechanic of rock breakage*. Pit and Quarry, Aug. 1963.

Ash, R.L.: *Design of blasting rounds*. Surface Mining, AIME, 1972.

Ash, R.L.: *Blasting characteristics of large diameter boreholes*. 6th Annual Drilling and Blasting Technology. Houston, 1977.

Dick, R.A. et al.: *Explosives and Blasting Procedures Manual*. US Bureau of Mines, 1983.

Hagan, T.N.: *The influence of controllable blast parameters on fragmentation and mining costs*. 1st, Symp. on Rock Fragmentation by Blasting. Lülea, 1983.

Hagan, T.N.: *Blast design considerations for underground mining and construction operations*. Design and Performance of Underground Excavations. ISRM/BGS. Cambridge, 1984.

Hagan, T.N.: *Increased profits through improved open pit blasting*. Proc. 4th Joint Meeting MMIJ-AIME. Tokyo, 1980.

Hemphill, G.B.: *Blasting Operations*. Mc Graw-Hill, 1981.

Konya, C.J. & D.R. Skidmore: *Blasthole depth and stemming height pressuring systems*. USBM, 1981.

Konya, C.J. & F. Otuonye: *Airblast reduction from effective blasthole stemming*. SEE, 1982.

Langefors, U. & B. Kihlstrom: *Voladura de Rocas*. Edit. URMO, 1973.

Lopez Jimeno, E.: *Parámetros Críticos en la Fragmentación de Rocas con Explosivos*. III Jornadas Minerometalúrgicas, Huelva, 1980.

Lopez Jimeno, C. & M. Abad: *Estudio de la Influencia del Seccionado de Cargas en un Mismo Barreno para Disminuir el Nivel de Vibraciones*. Canteras y Explotaciones, Noviembre, 1986.

Lopez Jimeno, E.: Implantación de un Método de Cálculo y Diseño de Voladuras en Banco. Tesis doctoral. ETS. Ing. de Minas de Madrid, 1986.

Melnikov, N.N.: *The Soviet Union. Recent and future development in surface coal mining*. CIM Bulletin. October, 1972.

Pla, F. et al.: *Curso de Perforación y Voladuras*. Fundación Gómez Pardo, 1978.

CHAPTER 20

Bench blasting

20.1 INTRODUCTION

In the preceding chapters, the influence of rock properties upon fragmentation, the criteria for selecting explosives and the relation of each blast parameter and its effect upon the results obtained, have been analized. What now has to be determined is the geometric configuration of the blastholes, the explosive charges, the initiation sequence and the delay timing, which constitute the main problems in blasting practice.

The extension of surface mining and the evolution of the drilling equipment have made bench blasting the most popular method of rock fragmentation with explosives. It has even been introduced and adapted for some underground operations.

The bench blastings for surface operations are classified according to their purpose. Below, the most common types are mentioned:

a) *Conventional bench blasting*. Pursuit of maximum fragmentation and swelling of the rock.

b) *Rip-rap blasting*. To obtain large fragments of rock.

c) *Cast blasting*. The use of explosives not only to fragment the rock but to also project a large quantity of it (overburden) to a pre-determined place, such as a spoil pile.

d) *Railway and highway construction blasting*. Conditioned by the road plan and the terrain.

e) *Trench and ramp blasting*. These are lineal operations where, due to the shape and narrowness of the excavations, the confinement of explosives is high.

f) *Ground leveling and foundation blasting*. These operations are usually over a small area and quite shallow.

g) *Preblasting*. This is to increase the natural fractures in the rock mass with little displacement.

This chapter will study the first three types of blastings.

Since the fifties, many formulas and methods to determine geometric parameters such as burden, spacing, subdrilling, etc. have been developed. These formulas use one or various groups of parameters: hole diameter, characteristics of explosives, compressive rock strength, etc. In Appendix I of this chapter, there is a résumé of the most important formulas for calculation.

Another common clasification for bench blastings is according to their blasthole diameters:

- *Small diameter blastings*, from 65 to 165 mm.
- *Large diameter blastings*, from 180 to 450 mm.

In small diameter blastings, the Swedish technique developed by Langefors and Kihlström can be followed, while in the larger it is better to use the crater technique established by Livingston or American criteria.

However, due to the great heterogeneity of the rocks, the method of calculation should be based upon a continous process of trial and error in order to arrive at conclusions.

In the following paragraphs some simple rules are given for an initial approach to the geometric design of blasting and the calculation of the charges, characterizing the rocks exclusively by their uniaxial compressive strength. It is obvious that in each case, after the tests and analysis of the initial results, it will be necessary to adjust the patterns and explosive charges according to the degree of fissurization and structural control exercised by the discontinuities existing in the rock mass.

20.2 SMALL DIAMETER BENCH BLASTING

Small diameter blasts are those that range between 65 and 165 mm drilling diameters and their principal applications are in quarries, construction excavations and small surface mining operations.

The explosive charges are elongated cylinders with a ratio $l/D > 100$, and they are usually composed of two types of explosives, one for the bottom charge and the other for the column charge.

20.2.1 *Drilling diameters*

The choice of blasthole diameter depends upon the average production per hour or rhythm of excavation and the compressive strength of the rock, Table 20.1.

It must be taken into account that drilling costs usually descend as the diameter increases.

20.2.2 *Bench height*

The height of the bench depends upon the loading equipment and the drilling diameter. The recommended dimensions, taking into account the scopes and characteristics of each group of machines, are found in Table 20.2.

For safety reasons, the maximum recommended height in mines and quarries is 15 m, and only for special applications, such as rip-rap blastings, should heights of 20 m be reached.

Table 20.1.

Blasthole diameter (mm)	Average production per hour (m^{3}b/h)	
	Medium-soft rock < 120 MPa	Hard-very hard rock >120 MPa
65	190	60
89	250	110
150	550	270

Table 20.2.

Bench height H (m)	Blasthole diameter D (mm)	Recommended loading equipment
8-10	65-90	Front end loader
10-15	100-150	Hydraulic or rope shovel

Photo 20.1. Bench blasting in a quarry.

Table 20.3.

Design parameter	Uniaxial compressive strength (MPa)			
	Low < 70	Medium 70–120	High 120–180	Very high > 180
Burden – B	39D	37D	35D	33D
Spacing – S	51D	47D	43D	38D
Stemming – T	35D	34D	32D	30D
Subdrilling – J	10D	11D	12D	12D

20.2.3 *Drilling, stemming and subdrilling patterns*

The burden B value depends upon the diameter of the blastholes, the rock characteristics and the type of explosives used. If the charge distribution is selective, with an explosive of high density and strength in the bottom and one of low density and medium strength in the column, the burden values oscilate between 33 and 39 times the blasthole diameter D, depending upon the compressive rock strength and the height of the bottom charge.

The spacing between blastholes in the same row varies between 1.15 B for hard rocks and 1.30 for soft rocks.

The stemming length and that of the subdrilling are calculated according to the blasthole diameters and the compressive rock strength.

In Table 20.3, the tentative values of the geometric parameters, as function of the compressive rock strength, are indicated.

20.2.4 *Inclination of the blastholes*

In the range of diameters mentioned, the drilling rigs are usually rotory percussive top hammer, pneumatic and hydraulic, or down-the-hole hammer. These machines permit inclinations of the feeds with angles of up to 20° or more with respect to the vertical.

The blasthole length L increases with inclination but, on the other hand, the subdrilling J decreases. To calculate L the following equation is used:

$$L = \frac{H}{\cos \beta} + \left(1 - \frac{\beta}{100}\right) \times J$$

where β represents the angle with respect to the vertical in degrees.

Table 20.4.

Design parameter	Compressive strength (MPa)			
	Soft < 70	Medium 70-120	Hard 120-180	Very hard > 180
Bottom charge length – l_f	30D	35D	40D	46D

20.2.5 *Charge distribution*

Taking into account the theory of selective charges, in which the energy per unit of length in the bottom of the blasthole should be from 2 to 2.5 times more than the energy required for rock breakage in front of the column charge and, in function with the compressive rock strength, Table 20.4 shows the recommended length for the bottom charges.

The height of the column charge is calculated by the difference between the length of the blasthole and the sum of the stemming length and that of the bottom charge.

The powder factor varies between 250 and 550 g/m^3 for the four rock groups shown.

20.2.6 *Example of application*

In a quarry, rock is extracted with a uniaxial compressive strength of 150 MPa in benches 10 m high. Drilling is carried out with a rotary percussive top hammer rig and a diameter of 89 mm. The explosives used are composed of a cartridged slurry of 75 mm in diameter and bulk ANFO with respective densities of 1.2 and 0.8 g/cm^3.

The drilling pattern and charge distribution are to be determined, maintaining the blasthole inclination at 20°.

- Subdrilling: $J = 12\,D = 1.1$ m
- Blasthole length

$$L = \frac{H}{\cos 20°} + \left(1 - \frac{20}{100}\right) \times J = 11.50\,\text{m}$$

- Burden $B = 35\,D = 3.1$ m
- Spacing $S = 43\,D = 3.8$ m
- Volume of broken rock

Photo 20.2. Multi-row bench blasting

$$VR = B \times S \times \frac{H}{\cos \beta} = 125.36 \text{ m}^3$$

- Yield of broken rock

$$RA = \frac{VR}{L} = 10.90 \frac{\text{m}^3}{\text{m}}$$

- Length of bottom charge $l_f = 40 \times D = 3.6$ m (It is considered that the weight of the column compresses the cartridges and their average diameter becomes 10% more than the nominal value).
- Concentration of the bottom charge $q_f = 6.4$ kg/m
- Bottom charge $Q_f = 23.0$ kg
- Length of the column charge $l_c = 5.1$ m
- Concentration of the column charge $q_c = 5.0$ kg
- Column charge $Q_c = 25.5$ kg
- Blasthole charge $Q_b = 48.5$ kg
- Powder factor

$$CE = \frac{Q_b}{VR} = 0.387 \text{ kg/m}^3$$

20.3 LARGE DIAMETER BLASTING

Blasts fired with hole diameters between 180 and 450 mm can be accounted for in this group. Drilling is usually carried out with rotary rigs and rolling cone bits which are applicable in large surface mining operations and in certain civil engineering excavations such as for power stations, quarries for the construction of dams, etc.

In this type of blasting, the design criteria was developed from Livingston's crater theory, with the cylindrical charges having a configuration such as to give $l/D < 50$.

20.3.1 *Drilling diameters*

As in small diameter blastings, the choice of this parameter is taken from the average production per hour and the type of rock to be fragmented, Table 20.5.

20.3.2 *Bench height*

The bench height is related to the reach of the rope shovels and the drilling diameter. According to the capacity of this loading equipment, the height in meters can be estimated with the following equation:

$$H = 10 + 0.57\,(C_c - 6)$$

where: C_c = the bucket size of the shovel (m^3).

Taking into account the compressive rock strength, the dimension of H can also be calculated from D with the average values indicated in Table 20.6.

In some cases, the bench height is limited by the geology of the ore deposit, due to imperatives of ore dilution control and safety measures, as indicated before. Generally speaking, in metal mining a relationship of $H/B < 2$ is maintained.

20.3.3 *Stemming*

The length of stemming is determined according to the diameter and compressive rock strength, Table 20.7.

20.3.4 *Subdrilling*

The subdrilling is usually calculated from blasthole diameter, Table 20.8.

When vertical blastholes are drilled, the subdrilling of the first row reaches values of 10 to 12 D.

Shorter subdrilling lengths than those indicated can be used in the following cases:

- Horizontal bedding planes that coincide with the bench toe.

Table 20.5.

Blasthole diameter (mm)	Average production per hour (m^3b/h)		
	Soft rock < 70 MPa	Medium hard 70-180 MPa	Very hard rock > 180 MPa
200	600	150	50
250	1200	300	125
311	2050	625	270

Table 20.6.

Design parameter	Compressive rock strength (MPa)		
	Low < 70	Medium-high 70-180	Very high > 180
Bench height – *H*	52 D	44 D	37 D

Table 20.7.

Design parameter	Compressive rock strength (MPa)		
	Low < 70	Medium-high 70–180	Very high > 180
Stemming – T	40 D	32 D	25 D

Table 20.8.

Design parameter	Blasthole diameter (mm)	
	180-250	250-450
Subdrilling – *J*	7-8 D	5-6 D

- Application of selective explosive charges.
- Use of inclined blastholes.

20.3.5 *Inclination*

In the range of diameters indicated it is quite frequent to use rotary drilling. Owing to the inconveniences that the angulation of the mast presents in this type of drills, especially in hard rock, vertical drilling is systematically used.

A typical example are the metal mining operations with bench heights of 10 to 15 meters.

However, in soft rocks and with bench heights over 24 meters, inclined drilling is recommended. This happens in side casting type coal mining operations.

20.3.6 *Drilling patterns*

The burden B value, as already indicated, is in function with the charge diameter, the compressive rock strength and the specific energy of the explosive used. The diameter of the column charge usually coincides with the drilling diameter, as it is quite normal to use bulk explosives with mechanized charging systems from trucks which, apart from faster filling, allow variations in the characteristics of the explosive along the length of the column.

In Table 20.9, the recommended burden and spacing values are indicated, in function with the type of rock and explosive used.

20.3.7 *Charge distribution*

In large surface operations, ANFO has been continually used as the only charge because of the following advantages:

- Low cost
- High Bubble Energy
- Safety
- Easy mechanization, etc.

The use of watergels has been limited to those cases where the use of ANFO was not possible such as, for example, when the blastholes had water inside or simply when the cartridges in the bottom acted as initiators or primers for the rest of the charge column.

At present, the development of emulsions and the possibility of mixing these with ANFO right in the charge truck (Heavy-ANFO) has given impulse to the use of selective charges.

The system consists in creating a bottom charge of a high density explosive with a length of 8 to 16 D, according to the rock type, and filling the rest of the blasthole with ANFO.

Table 20.9.

Type of explosive	Design parameter	Compressive rock strength (MPa)		
		Soft < 70	Medium hard 70-180	Very hard > 180
ANFO	Burden – B	28 D	23 D	21 D
	Spacing – S	33 D	27 D	24 D
Watergels/ emulsions	Burden – B	38 D	32 D	30 D
	Spacing – S	45 D	37 D	34 D

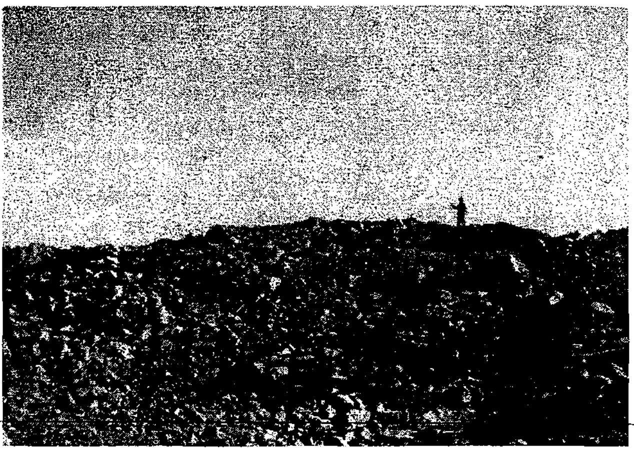


Photo 20.3. Marking of the mineral and the waste after a large diameter blast.

This charging technique gives minimum costs in drilling and blasting, along with optimum results in fragmentation, swelling, floor conditions and geometry of the muckpile.

In large diameter blastings, the powder factors vary between 0.25 and 1.2 kg/m^3.

20.3.8 *Application example*

In metal ore deposits, the blastings are drilled in a diameter of 251 mm with vertical blastholes, using two types of explosives, an emulsion for the bottom with a length of 8 D and a density of 1.3 g/cm^3 and the rest bulk ANFO with a density of 0.8 g/cm^3.

Calculate the patterns and the explosive charges knowing that the bench height is $H = 12$ m and the compressive rock strength is $RC = 110$ MPa.

- Subdrilling $J = 8\,D = 2.0$ m
- Length of blasthole $L = H + J = 14.0$ m
- Stemming $T = 32\,D = 8.0$ m
- Burden $B = 23\,D = 5.8$ m
- Spacing $S = 27\,D = 6.8$ m
- Breakage volume $VR = B \times S \times H = 473.3$ m^3
- Yield of the breakage $RA = VR/\mathrm{L} = 33.8$ m^3/ml
- Length of bottom charge $l_c = 8\,D = 2.0$ m
- Concentration of bottom charge $q_f = 64.24$ kg/m
- Bottom charge $Q_f = 128.5$ kg
- Length of the column charge $l_c = 4.0$ m
- Concentration of the column charge $q_c = 39.53$ kg/m
- Column charge $Q_c = 158.1$ kg
- Blasthhole charge $Q_b = 286.6$ kg
- Powder factor

$$CE = \frac{Q_b}{VR} = 0.605\ \mathrm{kg/m^3}$$

20.4 BENCH BLASTING WITH HORIZONTAL BLASTHOLES

In conventional bench blasting, the cut of the rock at floor level is achieved by subdrilling and by the concentration of high strength explosive in the bottom of the vertical blastholes. Even though this practice gives quite good results, there are cases in which the changing conditions of the rock masses make the cutting of the rocks in the lower part of the benches more difficult. In these situations the drilling length can increase as well as the height of the bottom charge, or the pattern can be complemented with horizontal blastholes or lifters. In central Europe, this blasting technique is quite widespread, due to its advantages in difficult rock masses:

- Improved cut of the rock at floor level.
- Lower concentration of explosives at bench bottom.
- Less fracturing on the ceiling of the underlying level.

On the other hand, the following disadvantages exist:

- Increase of specific drilling.
- Special device on the drilling rig to be able to drill horizontally.
- More moving of the drilling rig between the two work levels.

Generally speaking, the blastholes are drilled with the same diameter, ranging from 89 to 110 m.

As to the drilling patterns, the vertical blastholes are drilled up a distance of 0.5 to 1*B* from the horizontal ones, which means that the theoretical burden in the horizontal blastholes becomes:

$$B_2 = 0.5 \div 1 \times B$$

having: B = Burden of the vertical blastholes (m), B_2 = Burden of the horizontal blastholes (m).

The spacing between the horizontal blastholes S_2, with respect to those of the vertical ones is usually:

$$S_2 = 0.5\,S$$

where: S_2 = Spacing between horizontal blastholes (m), S = Spacing between the vertical blastholes (m).

The length of the horizontal blastholes H_2 depends upon the width of the blast, so it will be a multiple value of the burden of the vertical blastholes:

$$H_2 = n\,B$$

where: n = The number of rows of vertical blastholes.

20.5 RIP-RAP PRODUCTION BLASTING

In certain surface operations such as the construction of sea dikes and dams made of rock, materials with very specific and variable size distributions are necessary. The larger rock, within those distribution curves, is called *rip-rap*.

The configuration of the blasting that produces large blocks is different from conventional bench blasting. Two basic objectives consist of getting an adequate cut at floor level and a clean separation along the plane formed by the blastholes with the minimum splitting of the rock in front of that plane, Fig. 20.2.

The guidelines to follow for the design of rip-rap blastings are:

- Maximum possible bench height within safety limits. Usually between 15 and 20 meters.
- Drilling diameters between 75 and 115 mm.
- Blasthole inclination between 5 and 10 degrees.
- Subdrilling $J = 10\,D$.
- Length of bottom charge of 55 D, with explosives that give a high density.
- Ratio between the burden and the spacing of B/S = 1.4 - 1.70. Sometimes values of 2 or more are used.
- Bottom charge powder factor in function with the uniaxial compressive rock strength:

$$> 650\ \mathrm{g/m^3}\ \text{for RC} > 100\ \mathrm{MPa}$$
$$< 500\ \mathrm{g/m^3}\ \text{for RC} < 100\ \mathrm{MPa}$$

- Intermediate stemming between the bottom charge and the column charge of approximately 1 m.
- Charge density in the plane of the cut:

$$> 500\ \mathrm{g/m^2}\ \text{for RC} > 100\ \mathrm{MPa}$$
$$< 250\ \mathrm{g/m^2}\ \text{for RC} < 100\ \mathrm{MPa}$$

- Decoupled column charge with a ratio between the blasthole diameter and the charge diameter of around 2.
- Stemming with a length of 15 D.
- Instantaneous initiation sequence in the whole row of blastholes.

With the indicated design criteria, the actual results obtained in a large number of blastings carried out in homogeneous rock are shown in Table 20.10.

Table 20.10.

Block weight (kg)	Percentage (%)	
	$RC < 100$ MPa	$RC > 100$ MPa
> 3000	30	50
1000-3000	20	25
50-200	25	15
Fines	25	10

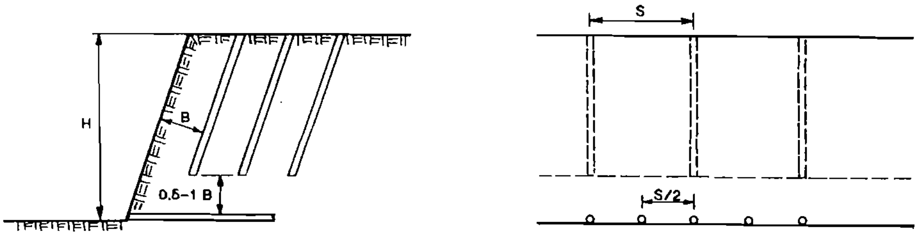


Fig. 20.1. Bench blasting with horizontal or lifter blastholes.

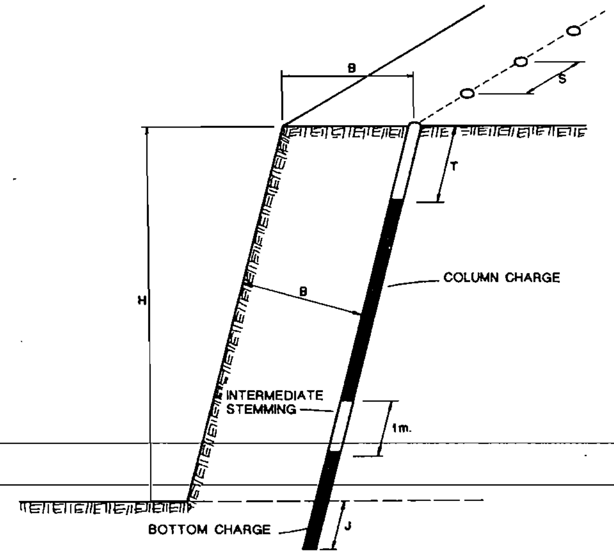


Fig. 20.2. Blasting pattern for rip-rap production.

Photo 20.4. Cast Blasting (Courtesy of Atlas Powder Co.)

20.6 CAST BLASTING

At the beginning of this decade, a blasting technique was introduced that not only fragmented rock but also cast a large volume of it into spoil piles and pits, Fig. 20.3.

The interest in this type of blasting is based on a reduced investment in machinery, operation and maintenance, as there is less manipulation of material due to the fact that a large volume of rock, from 40 to 60%, can be projected.

This method is being applied with great success in coal mines in South Africa, United States, Australia and Canada, but it can also be used in deposits of other ores with a horizontal or moderately inclined configuration.

According to Chironis (1982), Bauer et al. (1983), Grippo (1984) and Lang (1987), two groups or categories of criteria must be observed in order to design this type of blast; the operative and those pertaining to the actual blast.

The following conditions should be undertaken for operative criteria:

– *Bench height.* A minimum height of 12 m is recommended, although a lesser height can also be used for cast blasting.

– *Pit width.* The width of the pit left by a former blasting, which is equal to the width of the round, should be between 1 and 1.25 times the height of the face.

– *Wall angle.* In order to optimize the results of the operations, the active face walls can be excavated vertically with presplitting blasts fired before. With this, the burden remains constant throughout the front row, all the blastholes can be drilled vertically, the rock mass can be drained to allow use of explosives such as ANFO which are less expensive, etc.

Presplitting has been carried out with blastholes of 251 mm with 3 m spacing in soft rock and 5 to 6 m in hard rock and without stemming.

– *Type of rock formation.* The type of rock to be displaced affects the blasting, as soft rock generally requires a larger powder factor than hard rock. Also, the structural properties of the rock mass should be taken into consideration.

When analyzing the actual blast criteria, the most important are:

– *Burden.* Varies directly with the type of rock and the geology of the rock mass. The optimum displacement results are obtained with *reduced burdens* between 0.9 and 1.35 m/(kg/m)$^{1/2}$, using superior values when a limited casting is desired.

– *Type of explosive.* The explosive is important not only for the energy it develops but for its actual yield. The ratio between burden and explosive energy governs the ejection velocity of material from the face and, as a consequence, the distance reached. Experimentally, this velocity has been calculated using the following equation:

$$V_0\,(\text{m/s}) = 1.14 \times \left[\frac{B\,(\text{m})}{(\text{Energy})^{1/3}}\right]^{-1.17}$$

where: Energy (kcal/m) = $0.07853 \times d^2 \times p_c \times$ PAP, having: d = Charge diameter (cm), p_c = Explosive density (g/cm^3), PAP = Absolute weight strength (cal/g).

The minimum recommended velocity is around 15 m/s. The maximum horizontal reach can be calculated by knowing the trajectory exit angle β with the following equations:

$$DM_1 = \frac{V_0^2 \times \sin 2\beta}{g} \text{ (floor level rock)}$$

$$DM^2 = V_o \times \cos\beta \left[\frac{V_o \times \sin\beta + \sqrt{(V_o \times \sin\beta)^2 \times 2gh}}{g}\right]$$

where: DM_1 = Distance travelled by a fragment along a horizontal line level with the height of the fragment above the pit floor (m), DM_2 = Total distance travelled by a fragment ejected from the face, accounting for the height above the pit floor (m).

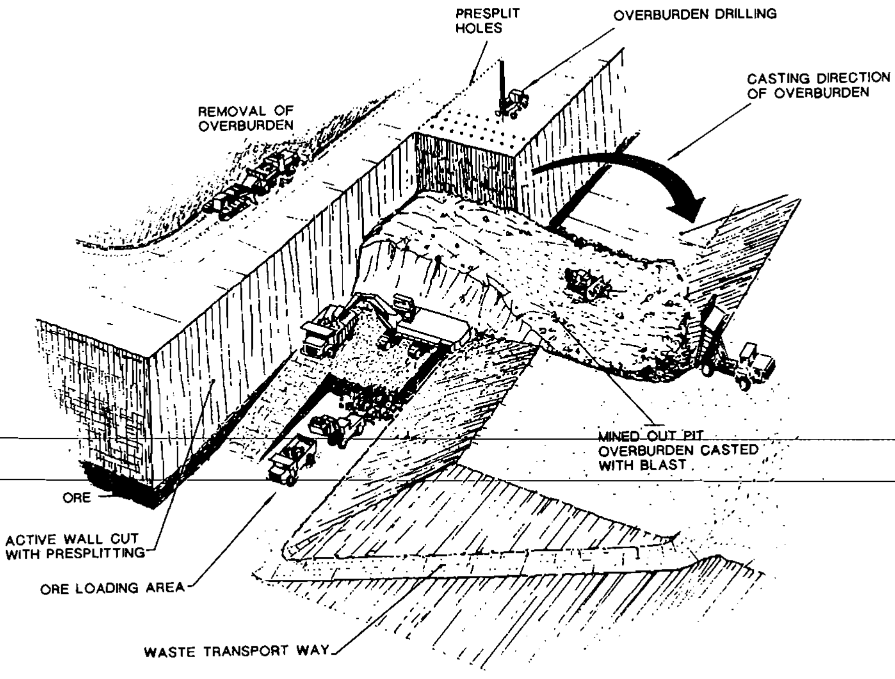


Fig. 20.3. Method of operation when blasting for maximum displacement.

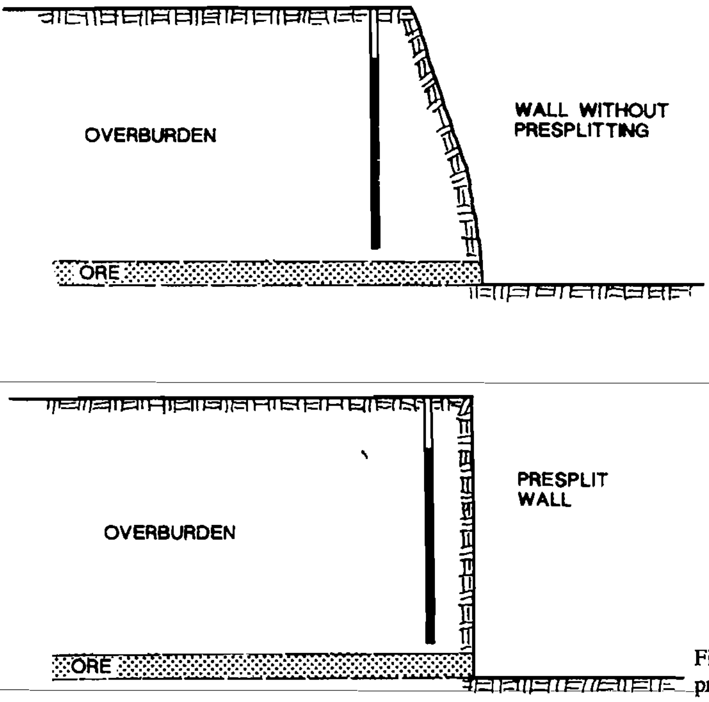


Fig. 20.4. Comparison between front row burden with and without presplitting.

– *Spacing.* The best results are obtained with rectangular staggered patterns and spacing between blastholes of 1.3 to 1.6 times the burden.

– *Stemming.* Depends upon the type of rock and the diameter of the blasthole. It is usually calculated to be between 20 and 25 *D*.

– *Subdrilling.* So as to not harm and dilute the ore, a negative subdrilling is used, filling the bottom of the blasthole above the contact to a length that oscilates between 4 and 6 *D*.

– *Initiation sequences and delay timing.* When only one face is involved, the initiation sequence is such that all the blastholes in the same row explode at the same instant and the delay is only between the rows themselves.

If the blast has two free faces, the sequence is usually in *V*1, Fig. 20.5.

The minimum delay times between blastholes are calculated with the practical rule of 20 to 35 ms/m of burden.

– *Powder factors.* They generally oscilate between 0.3 and 0.8 kg/m^3 for an ANFO type explosive.

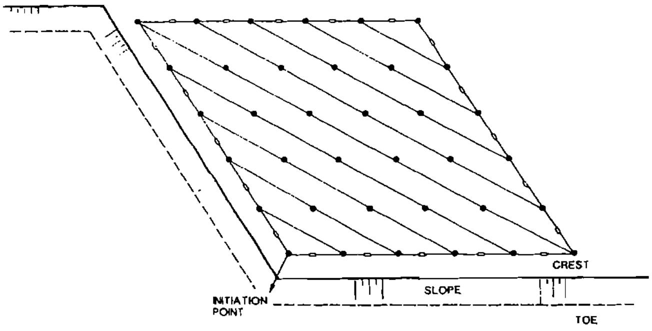


Fig. 20.5. Initiation sequence *V*1 for a staggered square pattern.

Table 20.11.

Rock blast-ability	Rock group classifica-tion	Compressive strength (MPa)	Strain energy factor (FE_l)	Strain factor (FV)
Tertiary	I	27	2.9	2.5
	II	30	2.9	2.5
	III	66	3.3	2.8
Cretaceous	II	21	2.8	2.8
	III	49	3.1	2.7
Pennsyl-vanian	VI	87	3.5	2.6
	VII	122	3.9	2.4
	VIII	108	3.7	2.5

– *Initiation.* It should be such as to maximize the Bubble Energy at the expense of the Strain Energy in order to better use the energy developed in rock displacement.

A method of calculating the blasts that produce maximum displacement has been developed by D'Appolonia Consulting Engineers. At first glance it looks complex, but it is simple to apply as only four nomographs and five equations are used.

The types of rock are classified by what is known as Strain Energy Factor and Blastability factor. Some examples are shown in Table 20.11.

For better comprehension, the method is applied to the following example of blasting:

- Drilling diameter D = 152 mm
- Strain Energy Factor FE_1 = 3

(This same value can be used if the rock characteristics are unknown.)

- Height of bench H = 7.5 m
- Desired throw distance of material DP = 1.8 m
- Density of Explosive $\rho_e = 0.67$ kg/m^3

The steps for calculating are:

1. Nomograph I. Draw a straight line to unite $FE_1 = 3$ with $DP = 18$ and the powder factor is obtained, $CE = 0.592$ kg/m^3.

2. Nomograph II. Draw the straight line that joins D = 152 mm and $\rho_e = 0.87$ kg, and the lineal concentration of explosive is determined, q_l = 15 kg/ml.

3. The values of C_1 and C_2 are calculated considering that K_1 and K_2 are equal to 1, which implies that, for the moment, the burden and spacing are equal:

$$C_1 = \frac{10.66 \times q_1}{CE \times K_2}$$

$$C_2 = \frac{0.3 \times K_1 \times C_1}{H}$$

4. Nomograph III. Knowing C_1 and C_2, C_3 is calculated as 1.400, and moving to the right of the nomograph making C_2' and C_3' equal to C_2 and C_3, respectively, the burden is calculated as B = 3.6 m.

5. The length of the charge in the blasthole is calculated.

$$l = H - K_1 \times B = 7.5 - 3.6 = 3.9 \text{ m}$$

6. Nomograph II. Draw the straight line which unites l = 3.9 m with q_l = 15 kg/m to obtain the total charge per blasthole Q_b = 68 kg.

7. Nomograph IV. Using this nomograph and the Table of Blastability Factors, the optimum burden can be calculated with $FE = 3$, $FV = 2.6$ and $B_o = 3.9$ m.

8. The values of B and B_o are compared. If the values are approximately equal, no other information is necessary to calculate the rest of the blasting parameters, because the stemming and spacing are determined with:

$$S = K_2 \times B$$
$$T = K_1 \times B$$

9. If B and B_o are not equal, as in this case, K_1 and K_2 are corrected by reducing them. D'Appolonia uses a rule of thumb which is $K_2 = K_l^3$, for another try. For this reason, if K_1 is chosen as 0.8, then K_2 = 0.51. These values are then used in the equations of C_1 and C_2. The process is repeated until B and B_o are equal.

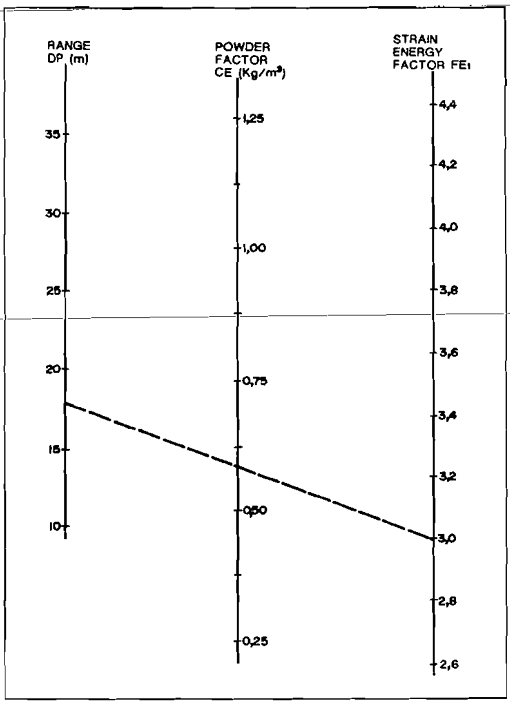


Fig. 20.6. Nomograph I.

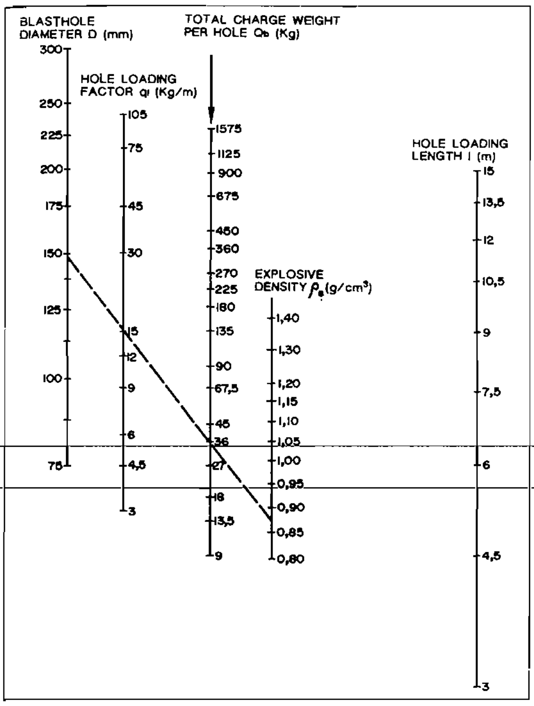
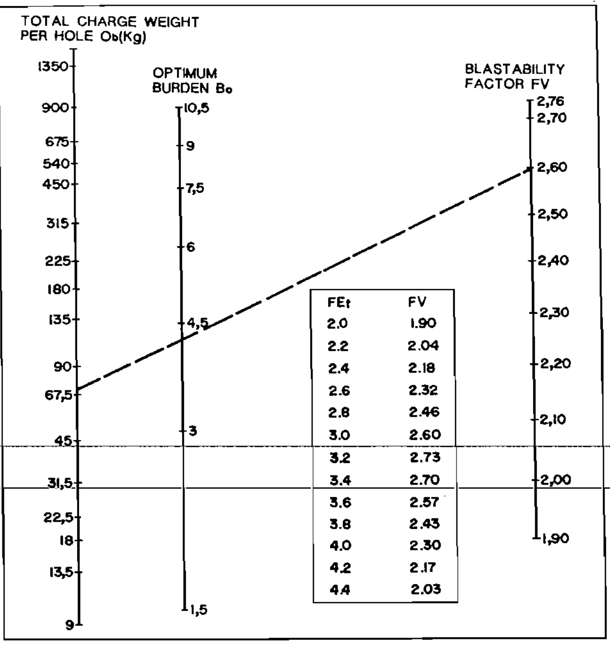


Fig. 20.7. Nomograph II.



FEt	FV
2.0	1.90
2.2	2.04
2.4	2.18
2.6	2.32
2.8	2.46
3.0	2.60
3.2	2.73
3.4	2.70
3.6	2.57
3.8	2.43
4.0	2.30
4.2	2.17
4.4	2.03

Fig. 20.9. Nomograph IV.

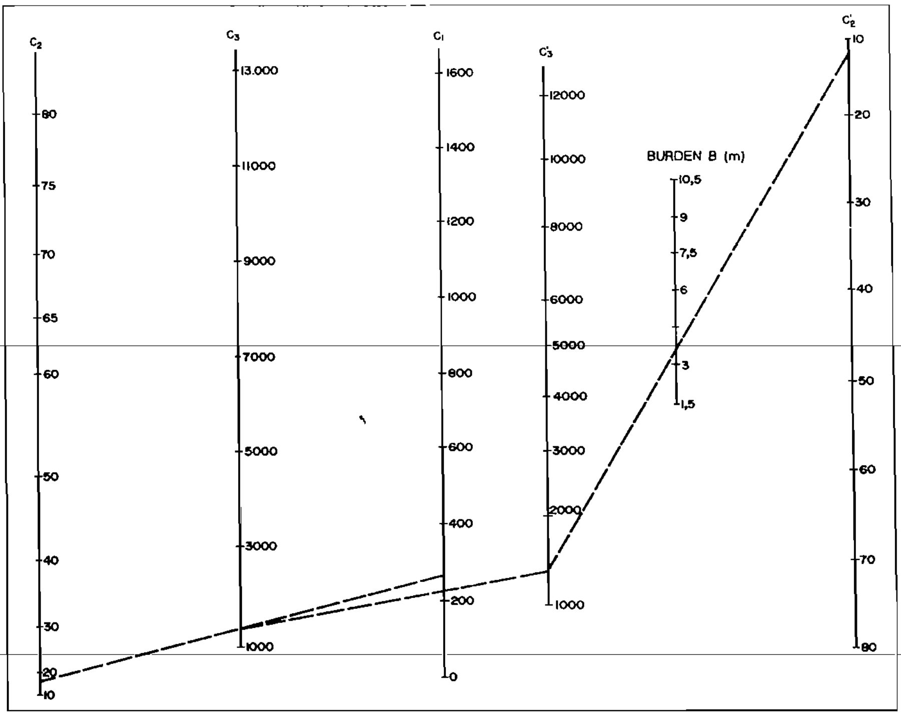


Fig. 20.8. Nomograph III.

APPENDIX 1: FORMULAS TO CALCULATE BENCH BLASTING PATTERNS

The burden, as indicated before, is the most critical geometric parameter in blast design. In order to determine the burden, during many decades, numerous investigations have been carried out and different methods of calculation have been developed.

In Table 20A.1, the most well known formulas for calculating the burden are shown, as explained in the following paragraphs, and the parameters that take part in each one.

The most complete formulas require knowing a large number of data

Table 20A.1. Matrix of the comparison of formulas to calculate burden value in bench blasting.

Parameters used	Andersen	Fraenkel	Pearse	Hino	Allsman	Ash	Langefors	Hansen	Ucar	Konya	Foldesi	Praillet	L.Jimeno	Berta	Carr	Konya	Olofsson	Rustan
Diameter of blasthole or of charge	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X
Bench height								X	X			X					X	
Length of blasthole	X	X																
Stemming												X						
Subdrilling												X						
Length of charge		X						X	X									
Inclination of blasthole							X							X			X	
Rock density				X	X					X			X		X	X		
Compressive rock strength or equivalent indexes		X	X	X							X	X		X				
Rock constants or factors	X		X				X	X									X	
Seismic velocity of the rock mass													X	X	X	X		
Density of the explosive						X	X	X	X	X	X	X	X	X	X		X	
Detonation velocity						X					X	X	X	X	X			
Detonation pressure			X	X	X													
Binomial rock-explosive constant				X														
Burden/spacing ratio							X											
Strength of explosive							X										X	
Loading equipment												X						

which, in most cases, are not exact because the actual field characteristics of the blasts change frequently and a global study is not worthwhile.

For this reason, the authors of this manual feel that in a near future, all the classic equations are going to be used as guidelines for designing tentative blastings and that afterwards, once the characteristics of the rocks have been determined through monitoring the blasthole drilling, the optimum patterns or the explosive charges in each blasthole for an established pattern, will be determined.

On the other hand, when explosives that differ in strength from those used under given conditions with an established pattern, the new burden value will be calculated with the following equation:

$$\left[\frac{\text{PRP of the new explosive}}{\text{PRP of the original explosive}}\right]^{1/3} \times \text{Burden} = \text{New burden}$$

Following this, the spacing will be determined keeping the same relationship S/B as in the original pattern.

As in the case of changing the drilling diameter, the new geometric pattern of the blast will be established from the new burden, obtained with the following:

$$\left[\frac{\text{Diameter of the new blastholes}}{\text{Diameter of the original holes}}\right]^{2/3} \times \text{Burden} = \text{New burden}$$

1. *Andersen (1952)*

$$B = K \times \sqrt{D' \times L}$$

B = Burden (feet), D' = Diameter (feet), L = Length of blasthole (feet), K = Empirical constant.

As in many cases good results were obtained by making $K = 1$, and by taking the diameter in inches, the former equation when used in practice, ended up as:

$$B = \sqrt{D \times L}$$

where: D = Blasthole diameter (inches).

This formula does not take into account the rock properties nor those of the explosive.

The burden value increases with the length of the blasthole but not indefinitely as usually happens in practice.

2. *Fraenkel (1952)*

$$B = \frac{R_v \times L^{0.3} \times l^{0.3} \times D^{0.8}}{50}$$

where: B = Burden (m), L = Length of blasthole (m), l = Length of charge (m), D = Blasthole diameter (mm), RV = Resistance to blasting which oscillates between 1 and 6 in function with rock type. Rocks with high compressive strength (1.5). Rocks with low compressive strength (5).

The following simplified relationships are usually put into practice:

- B is reduced to $0.8\ B < 0.67\ L$.
- l is taken as $0.75\ L$.
- S should be less than $1.5\ B$.

3. *Pearse (1955)*

Using the concept of the deformation energy per unit of volume, Pearse obtained the following equation:

$$B = K_v \times 10^{-3} \times D \times \left[\frac{PD}{RT}\right]^{1/2}$$

where: B = Maximum burden (m), K_v = Constant that depends upon the rock properties (0.7 to 1.0), D = Blasthole diameter (mm), PD = Detonation pressure of the explosive (kg/cm^2), RT = Tensile strength of the rock (kg/cm^2).

4. *Hino (1959)*
The calculation equation proposed by Hino is:

$$B = \frac{D}{4}\left(\frac{PD}{RT'}\right)^{1/n}$$

where: B = Burden (m), D = Blasthole diameter (cm), PD = Detonation pressure (kg/cm^2), RT' = Dynamic tensile strength (kg/cm^2), n = Characteristic constant depending upon the par explosive-rock and calculated through the cratering test.

$$n = \frac{\log \dfrac{PD}{RT'}}{\log 2 \dfrac{D_o}{d/2}}$$

where: D_o = Optimum depth of the center of gravity of the charge (cm) determined graphically from the following equation values:

$$D_g = \Delta \Sigma V_e^{1/3}$$

where: d = Diameter of the explosive charge, D_g = Depth of the center of gravity of charge, Δ = Relationship of depths D_g/D_c, D_c = Critical depth to center of gravity of charge, Σ = Volumetric constant of charge, V_e = Volume of the charge used.

5. *Allsman (1960)*

$$B_{max} = \sqrt{\frac{\text{Impulse} \times g}{\pi \times \rho_r \times u}} = \sqrt{\frac{PD \times D \times \Delta t \times g}{\rho_r \times u}}$$

where: B_{max} = Maximum burden (m), PD = Mean adverse detonation pressure (N/m^2), Δt = Duration of the average detonation (s), π = 3.1416, ρ_r = Specific rock weight (N/m^3), u = Minimum velocity which must be imparted to the rock (m/s), D = Blasthole diameter (m), g = Gravity acceleration (9.8 m/s^2).

6. *Ash (1963)*

$$B(\text{ft}) = \frac{K_B \times D(\text{in})}{12}$$

Photo 20.5. Results of a single-row blast in a 20 m high bench.

Table 20A.2.

Type of explosive	Rock group		
	Soft	Medium	Hard
Low density (0.8 to 0.9 g/cm^3) and low strength	30	25	20
Medium density (1.0 to 1.2 g/cm^3) and medium strength	35	30	25
High density (1.3 to 1.6 g/cm^3) and high strength	40	35	30

Table 20A.3.

Type of rock	F_r (kg/m^3)	RC (MPa)	RT (MPa)
I	0.24	21	0
II	0.36	42	0.5
III	0.47	105	3.5
IV	0.59	176	8.5

where K_B depends upon the rock group and the type of explosive used, Table 20A.2.

- Blasthole depth L = $K_L \times B$ (K_L between 1.5 and 4)
- Subdrilling J = $K_J \times B$ (K_J between 0.2 and 0.4)
- Stemming T = $K_T \times B$ (K_T between 0.7 and 1)
- Spacing S = $K_s \times B_2$ K_s = 2.0 for simultaneous initiation.

K_s = 1.0 for sequenced blastholes with long delay.
K_s = between 1.2 and 1.8 for sequenced blastholes with short delay.

7. *Langefors (1963)*
Langefors and Kihlström propose the following equation to calculate the Maximum Burden value B_{max}.

$$B_{max} = \frac{D}{33}\sqrt{\frac{\rho_e \times \text{PRP}}{\bar{c} \times f \times (S/B)}}$$

where: B_{max} = Maximum burden (m), D = Diameter in the bottom of the blasthole, $\bar{c}$ = Rock constant (calculated from C), f = Degree of fixation (vertical blastholes f = 1, inclined blastholes 3:1 f = 0.9, inclined blastholes 2:1 f = 0.85), S/B = Spacing/burden ratio, ρ_e = Loading density (kg/dm^3). PRP = Relative weight strength of the explosive (1-1.4).

The constant c is the quantity of explosive necessary to fragment 1 m^3 of rock, normally in surface blasts and with hard rock c = 0.4 is taken. This value is modified in accordance with:

B = 1.4 – 15 m, $\bar{c} = c + 0.75$
B < 1.4 m, $\bar{c} = 0.07/B + c$

The practical burden is determined from:

$$B = B_{max} - e' - d_b \times H$$

where: H = Bench height (m), e' = Collaring error (m/m), d_b = Blasthole deviation (m).

8. *Hansen (1967)*
Hansen modified the original equation proposed by Langefors and Kihlström, arriving at the following:

$$Q_b = 0.028\left(\frac{H}{B} + 1.5\right) \times B^2 + 0.4 \times F_r\left(\frac{H}{B} + 1.5\right) \times B^3$$

where: Q_b = Total explosive charge per blasthole (kg), H = Bench height (m), B = Burden (m), F_r = Rock factor (kg/m^3)

The rock factors F_r are determined by using the Table 20A.3.

9. *Ucar (1972)*
The equation developed by Ucar is:

$$1.5 \times B^2 H + 2B \times q_l - 3H \times q_l = 0$$

where: B = Burden (m), H = Bench height (m), q_l = Charge concentration (kg/m).

The value of B is obtained by solving the former second degree equation.

This author starts from the following hypotheses:

Powder factor (0.4 kg/m^3).
Total explosive charge per blasthole (kg)
$Q_b = 0.4 \times B \times S \times H$.
Lineal charge concentration (kg/m)
$q_l = \rho_e \times (D/36)^2$.
Length of charge (m) $l = H - B + B/3$.
Spacing equal to burden.

where: ρ_e = Density of explosive (g/cm^3), D = Charge diameter (mm), S = Spacing (m).

10. *Konya (1972)*

$$B = 3.15 \times D \times \left[\frac{\rho_e}{\rho_r}\right]^{0.33}$$

where: B = Burden (feet), D = Diameter of explosive (inches), ρ_e = Specific gravity of explosive, ρ_r = Specific gravity of rock.

The spacing is determined from the following equations.

Instantaneous single-row blastholes.

$$H < 4B, S = \frac{H + 2B}{3}$$

$$H > 4B, S = 2B$$

Sequenced single-row blastholes.

$$H < 4B, S = \frac{H + 7B}{8}$$

$$H > 4B, S = 1.4B$$

Stemming
Massive rock $T = B$
Stratified rock T = 0.7B

11. *Földesi (1980)*

The Hungarian method for calculation proposed by Földesi and his collaborators is the following:

$$B = 0.88 \times D \times \sqrt{\frac{\rho_e}{m \times CE}}$$

where: B = Burden (m), D = Blasthole diameter (mm), ρ_e = Specific densith of the explosive in the blasthole (kg/m^3), CE = Powder factor (kg/m^3).

$$m = 1 + \frac{0.693}{\ln(\rho_e \times VD^2) - \ln RC - 1.39}$$

where: VD = Detonation velocity of explosive (m/s), RC = Compressive rock strength (MPa).

In instantaneous initiation take $2.2 < m < 2.8$, and for sequences with delay initiations take $1.1 < m < 1.4$.

Other parameters are:

Spacing $S = m \times B$
Distance between rows $B_f = 1.2 \times B$

$$\text{Stemming } T = 1.265 \times \frac{B \times VD}{VC} \times \sqrt{\frac{\rho_e}{\rho_s}}$$

where: ρ_s is the density of the stemming material in the blasthole.

Subdrilling $J = 0.3 \times B$

12 *Praillet (1980)*

Based on the formula of Oppenau, Praillet proposes the following equation to calculate B.

$$B^3 + \frac{B^2 \times (H \times K)}{D} - \left[\frac{2.4 \times \rho_e \times \left[\frac{VD}{4000}\right]^2 \times (H + J - T) \times D^2}{10 \times RC}\right] = 0$$

where: B = Burden (m), $S = B$, H = Bench height (m), K = Constant (12.5 for rope shovel and 51 for dragline), ρ_e = Specific gravity of explosive, VD = Detonation velocity of explosive (m/s), J = Subdrilling (m), T = Stemming (m), D = Blasthole diameter (mm), RC = Compressive strength of the rock (MPa).

The value of B cannot be determined directly and it is necessary to have a micro computer to calculate it by successive approximations.

13. *Lopez Jimeno, E. (1980)*

Modifies Ash's formula by incorporating the seismic velocity to the rock mass, resulting in:

$$B = 0.76 \times D \times F$$

where: B = Burden (m), D = Diameter of blasthole (inches), F = Correction factor based on rock group and type of explosive. $F = fr \times fe$.

$$fr = \left[\frac{2.7 \times 3500}{\rho_r \times VC}\right]^{0.33}$$

$$fe = \left[\frac{\rho_e \times VD}{1.3 \times 3660^2}\right]^{0.33}$$

where: ρ_r = Specific gravity of rock (g/cm^3, VC = Seismic propogation velocity of the rock mass (m/s), ρe = Specific gravity of explosive charge (g/cm^3), VD = Detonation velocity of explosive (m/s).

The indicated formula is valid for diameters between 165 and 250 mm. For larger blastholes the burden value will be affected by a reducing coefficient of 0.9.

14. *Konya (1983)*

$$B = \left[\frac{2\rho_e}{\rho_r} + 1.5\right] \times d$$

where: B = Burden (feet), ρ_e = Specific gravity of explosive, ρ_r = Specific gravity of rock, d = Diameter of explosive (inches).

Other design parameters determined from the Burden are:

- Spacing (feet)
- Instantaneous initiation single-row blastholes

$$\text{Low benches: } H < 4B, \quad S = \frac{H + 2B}{3}$$

$$\text{High benches: } H \geq = 4B, \quad S = 2B$$

- Delayed initiation single-row blastholes

$$\text{Low benches: } H < 4B, \quad S = \frac{H + 7B}{8}$$

$$\text{High benches: } H \geq = 4B, \quad S = 1.4B$$

- Stemming (feet) $T = 0.7B$
- Subdrilling (feet) J = $0.3B$

15. *Berta (1985)*

The formula used by this author is:

$$B = d\sqrt{\frac{\pi \times \rho_e}{4 \times CE}}$$

where: B = Burden (m), d = Charge diameter (m), ρ_e = Density of explosive (kg/m^3), CE = Powder factor (kg/m^3).

To determine *CE*, the following equation is used:

$$CE = \frac{g_f \times \varepsilon_s}{n_1 \times n_2 \times n_3 \times \varepsilon}$$

having: g_f = New surface created by crushing, referred to as the unit of rock volume (m^2/m^3). Taking that $g_f = 64/M$, where M is the maximun dimension of the blast material. ε_s = Specific surface energy (MJ/m^2), ε = Energy per unit of mass (MJ/kg), n_1 = Impedance factor, n_2 = Coupling factor, n_3 = Breaking factor, normally 0.15.

At the same time, the values of n_1 and n_2 and calculated from:

$$n_1 = \frac{(\rho_e \times VD - \rho_r \times VC)^2}{(\rho_e \times VD + \rho_r \times VC)^2}$$

and

$$n_2 = \frac{1}{e^{D/d} - (e-1)}$$

where: VD = Velocity of detonation (m/s), VC = Seismic velocity (m/s), ρ_r = Density of the rock (kg/m^3), D = Blasthole diameter (m).

16. *Bruce Carr (1985)*

The method proposed by Carr includes the following calculations:

- Characteristic impedance of rock

$$Z_r = 1.31 \times \rho_r \times \frac{VC}{1.000}$$

where: ρ_r = Specific gravity of the rock, VC = Sonic velocity of the rock (feet/s).

- Detonation pressure of the explosive:

$$PD = \frac{0.418 \times \rho_e \times \left[\frac{VD}{1.000}\right]^2}{0.8 \times \rho_e + 1}$$

having: ρ_e = Density of explosive, VD = Detonation velocity of explosive (feet/s).

- Characteristic powder factor

$$CEC = \frac{Z_r}{PD}$$

- Spacing between blastholes

$$S = 3\sqrt{\frac{\rho_e \times d^2}{CEC}}$$

where: d = Charge diameter (inches).

- Burden $B = S \times 0.833$
- Stemming $T = B$
- Subdrilling $J = (0.3\text{-}0.5) \times S$

17. *S.O. Olofsson (1990)*

Olofsson offers the following simplified equation based on Langefors' formula:

$$B_{max} = K \times \sqrt{q_f} \times R_1 \times R_2 \times R_1$$

where: K = Constant that depends upon the type of explosive; Gelatin-like explosives 1.47; Emulsions 1.45; ANFO 1.36. q_f = Concentration of the bottom charge of the chosen explosive (kg/m), R_1 = Side inclination correction factor, R_2 = Rock type correction factor, R_3 = Bench height correction factor.

The correction factors R_1 and R_2 are determined for the different work conditions with the following tables:

Table 20A.4.

Inclination	∞:1	10:1	5:1	3:1	2:1	1:1
R_1	0.95	0.96	0.98	1.00	1.03	1.10

Table 20A.5.

Rock constant 'C'	0.3	0.4	0.5
R_2	1.15	1.00	0.90

When the bench heights satisfy $H < 2B_{max}$ and the drilling diameters are smaller than 103 mm, the value of R_3 is obtained with the equation:

$$R_3 = 1.16 - \left[0.16\,\frac{H_2}{H_1}\right]^2$$

where: H_1 = Actual bench height, H_2 = Bench height $= 2B_{max}$ ($H_2 > H_1$).

To calculate the practical burden, the same formula as in Langefors method is applied.

18. *Rustan (1990)*

The equation for the burden of surface mines is:

$B = 18.1 \cdot D^{0.689}$ (+ 52% maximum expected
value and – 37% minimum value)

where: D = Blasthole diameter (between 89 and 311 mm). This formula was obtained by regression analysis from a 73 statistic data, with a corelation coefficient of $r = 0.78$.

For underground mines, taken from 21 field examples, the burden formula is:

$B = 11.8 \cdot D^{0.630}$ (+ 40% maximum expected
value and – 25% for the minimum value)

where: D = Blasthole diameter (between 48 and 165 mm) and the correlation coefficient $r = 0.94$.

REFERENCES

Anonymous: *Casting overburden: An explosive idea.* Coal Age, July 1982.

Ash, R.L.: *The design of blasting rounds.* Chapter 7.3. Surface Mining, AIME, 1968.

Berta, G.: *L'Explosivo Strumento Di Lavoro.* Italexplosivi, 1985.

Bauer, A. et al.: *Principles and applications of displacing overburden in strip mines by explosive casting.* SME, AIME, October, 1983.

Chernigouskii, A.A.: *Application of directional blasting in mining and civil engineering.* United States Department of the Interior and the National Science Foundation, 1985.

Chironis, N.P.: *Casting overburden by blasting.* Coal Age, May, 1980.

Chironis, N.P.: *Mines take on explosive casting.* Coal Age. October, 1981.

Chironis, N.P.: *Blast-casting pays off at Avery.* Coal Age. November, 1982.

Chironis, N.P.: *Angled drilling aids casting.* Coal Age, January, 1984.

Chironis, N.P.: *Spiced-up ANFO mixed leads to super blast for casting.* Coal Age. May, 1984.

Chironis, N.P.: *Blast casting succeeds at multi-seam western mine.* Coal Age. November, 1985.

Gregory, C.E.: *Explosives for North American engineers.* Trans Tech Publications. Third Edition.1983,

Grippo, A.P.: *How to get more cast per blast.* E/MJ. December, 1984.

Hagan, T.N.: *Optimum design features of controlled trajectory.* SEE, 1979.

Hempfill, G.B.: *Blasting Operations.* Mc Graw-Hill, 1981

Konya, C.J.: *Blasting Design.* Surface Mining Environmental Monitoring and Reclamation Handbook. 1983.

Langefors, U. & B. Kihlstrom: *Voladura de Rocas.* Edit. Urmo, 1973.

Lopez Jimeno, E.: *Parámetros Críticos en la Fragmentación de Rocas con Explosivos.* VI Jornadas Minerometalúrgicas, Huelva, 1980.

Lopez Jimeno, E. & C. Lopez Jimeno: *Las Voladuras en Cráter y su Aplicación a la Minería.* VI Congreso Internacional de Minería y Metalurgia, Barcelona, 1984.

Lopez Jimeno, E.: Implantación de un Método de Cálculo y Diseño de Voladuras en Banco. Tesis Doctoral. E.T.S. de Ingenieros de Minas de Madrid, 1986.

Grönfors, T. & L. Vajda: *Modern equipment and calculation methods for stone winning and rock breaking.* Surface Mining and Equipment, IMM. 1983.

Ludwiczak, J.T.: *The Blasting Primer.* B.M.C., 1984.

McDonald, K.L. et al.: *Productivity improvements for dragline operations using controlled blasting in a single and multiple seam opencast voal operation, Rietspruit, South Africa.* March, 1982.

Olofsson, S.O.: *Applied explosives technology for construction and mining.* Applex, 1990.

Paveto, C.S.: *Surface Mine Blasting,* 1986.

Petrunyak, J. & C. Postupack: *Explosive energy is challenging mechanical energy for overburden removal.* Coal Mining and Processing. July, 1983.

Praillet, R.: *A New Approach to Blasting.*

Pugliese, J.M.: *Designing blast patterns using empirical formulas.* US Bureau of Mines. I.C. 8550, 1972.

Rustan, P.A.: *Burden, spacing and borehole diameter at rock blasting.* The Third International Symposium on Rock Fragmentation by Blasting. 1990

Speath: *Formula for the proper blasthole spacing.* Engineering News. Record, 218, 1960.

Thomson, C.D.: *Correcting the burden formula for heave blasting.* Mining Engineering.

Tamrock: *Handbook on Surface Drilling and Blasting.* 1984.

Ucar, R.: *Importancia del Retiro en el Diseño de Voladuras y Parámetros a Considerar.* GEOS, 24-3-10, Caracas, Enero, 1978.

Viera, M.: *Sensitivity analysis, a key optimum blasting tunnels.* 18th APCOM, 1984.

CHAPTER 21

Blasting in other surface operations

21.1 INTRODUCTION

In this chapter, other types of surface excavations that require explosives will be briefly studied.

There is a combination of operations that are characterized by their great length and by the changing conditions of geometry and properties of the rock to be broken, which impose the design of the projects and the profile of the ground. Such is the case in excavations for highways and railways, as well as for pipelines.

Finally, the blastings to open ramps, for ground levelling and foundations, and for swelling of rocks or preblastings, will be discussed.

21.2 EXCAVATIONS FOR HIGHWAYS AND RAILWAYS

The rock excavations done by blasting in highway and railway construction are of two different types: through cut (1) and side hill cut (2), Fig.21.1.

In both cases, the tendency is to do the excavation in one shot, however when the cuts are deep (> 15 m), it is sometimes necessary to carry out the operation in two phases. Other factors that can influence the operation are:

- Safety.
- Limits in noise, airblast and vibration levels.
- Advance velocity.
- Capacity of the loading equipment, etc.

Due to the importance of rock condition in the remaining walls, especially those of elevated height, it is normal to finish the excavation with contour blastings, which is another reason to keep the depth of cut from 10 to 12 m, in order to maintain drilling precision.

21.2.1 *Drilling diameters*

Normally, small diameter blastholes are used because of the following advantages:

- Better pattern adaptation to irregular ground profiles.
- Good rock fragmentation due to better distribution of the explosives. This enables the loading to be carried out with less equipment.
- Lower vibration and airblast levels.
- Possibility of contracting drilling and blasting teams, and
- Less damage in remaining rock which lowers the cost of grouting and supporting.

The usual diameters vary between 65 and 125 mm. It is not unusual to carry out the production blasts with calibers between 89 and 125 mm and the contour blastings between 65 and 75 mm.

Apart from small sections which can be drilled with handheld drills and are used as work platforms for subsequent phases, the drilling rigs are usually crawlers with top hammer.

As the drilling diameter is influenced by the bench height, in through cuts the blastholes may be larger than in side hill cuts. In general, the following relationship should exist:

$$D = H/60$$

where: D = Blasthole diameter, H = Excavation depth.

21.2.2 *Drilling lengths*

The blasthole lengths depend upon the bench height, the inclination, which is usually 15 to 20 degrees, and the necessary subdrilling which depends upon the compressive rock strength:

$$L = \frac{H}{\cos \beta} + \left[1 - \frac{\beta}{100}\right] \times J$$

where: β = Angle with respect to vertical in degrees, H = Bench height (m), J = Subdrilling, estimated from Table 21.1.

21.2.3 *Charge distribution and stemming*

In this type of blasting, selective explosive columns are used. The bottom charges are gelatin dynamites or slurries, and ANFO is used in the column charges.

Table 21.2 shows the recommended lengths for bottom charges and stemming in different types of rock. The heights of the column charges are calculated by the difference between the length of the blastholes and the sum of the bottom charges and stemmings.

21.2.4 *Drilling patterns*

Trench cuts

These are always carried out with vertical blastholes and,

Table 21.1.

Design parameter	Uniaxial compressive strength (MPa)			
	Low	Medium	High	Very High
	< 70	70-120	120-180	> 180
Subdrilling – J	10 D	11 D	12 D	12 D

Table 21.2.

Design parameter	Uniaxial compressive strength (MPa)			
	Low	Medium	High	Very high
	< 70	70-120	120-180	> 180
Bottom charge length – l_f	30 D	35 D	40 D	46 D
Stemming – T	35 D	34 D	32 D	30 D

Table 21.3.

Design parameter	Uniaxial compressive strength (MPa)			
Design parameter	Low	Medium	High	Very high
	< 70	70-120	120-180	> 180
Burden – B	39 D	37 D	35 D	33 D
Spacing – S	51 D	47 D	43 D	38 D

Table 21.4.

Design parameter	Uniaxial compressive strength (MPa)			
	Low	Medium	High	Very high
	< 70	70-120	120-180	> 180
S/B – Ratio	1.25	1.20	1.15	1.15
Powder Factor – CE (kg/m^3)	0.30	0.35	0.42	0.49

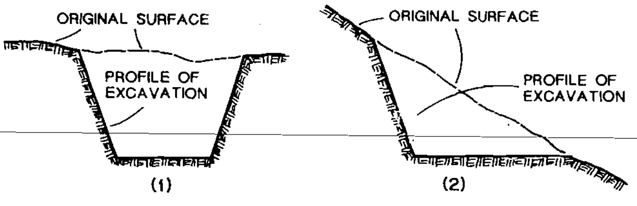


Fig. 21.1. Excavations with through (1) and side hill (2) cuts.

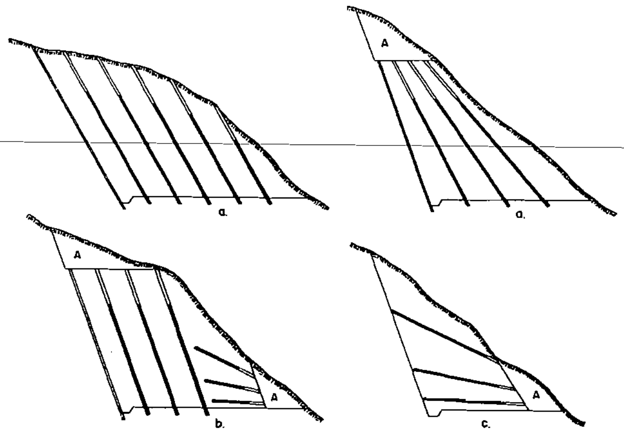


Fig. 21.2. Side hill cut excavations

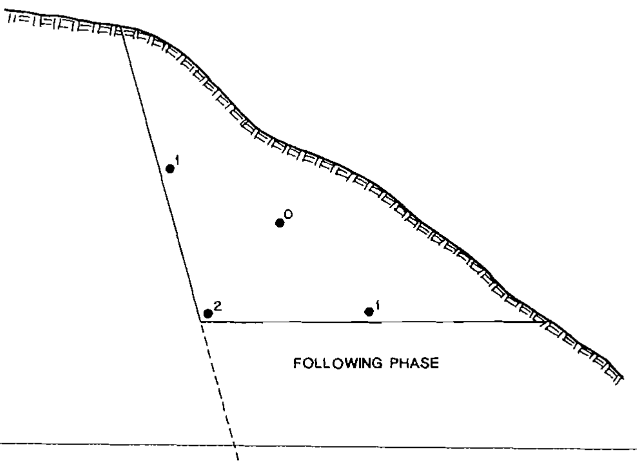


Fig. 21.3. Opening of access ways for posterior side hill cuts.

according to the *H/D* relationship, two different cases can be distinguished.

a) $H > 100\ D$. This is the most common in bench heights of 10 to 12 m. The burden and spacing values are calculated from Table 21.3.

b) $H < 100\ D$. In these cases the burden is calculated from the equation:

$$B = \left[\frac{Q_b}{\frac{S}{B} \times \frac{H}{\cos\beta} \times CE} \right]^{0.5}$$

where: Q_b = Total charge per blasthole (kg), H = Bench height (m), S/B = Relationship between Spacing and Burden (Table 21.4), CE = Powder factor (Table 21.4), β = Angle with respect to the vertical in degrees.

Side hill cuts

This type of operation can be carried out in three different ways.

a) Vertical blastholes, either parallel or in fan cut.

b) Vertical and horizontal blastholes.

c) Horizontal blastholes or lifters.

The opening of the access ways pointed out in Fig. 21.2 with the letter *A* are usually carried out with the same drilling equipment that will be used for production drilling, although with a smaller diameter; using horizontal blastholes that are parallel to the profile, in sufficient number to open platforms with a width of between 6 and 9 meters. The work cycle is discontinuous because after each round it is necessary to take away the broken rock, usually with a crawler tractor, to enable drilling of the next blast.

Charging with cartridge explosives is hard work and for this reason, if water is not present, pneumatic ANFO chargers are used.

The stemming lengths are determined according to Table 21.2. Clay plugs may also be used to facilitate the execution and effectivity.

Lifter blastings have the following advantages:

- They require minimum preparation.

- Rock breakage costs are low due to the benefit of the rock that is detached by effect of gravity.

However, there are serious problems which do not make them recommendable for large projects:

- Large quantity of flyrock as the charges act as incrater blasting.
- The remaining mass is left quite deteriorated with hanging rocks and, on occasions, with inverted walls.

If drilling is done vertically, apply the values indicated for the relationship $H < 100\,D$.

When horizontal or lifter holes are used, the calculation for the pattern will be based upon the equation:

$$S = 3 \times \sqrt{D \times L}$$

where: S = Spacing (m), D = Blasthole diameter (m), L = Blasthole length (m).

If the bench height is under 5 m, a single-row of blastholes will be used, between 5 and 8 m two rows, and, for more than 8 m, three or more rows.

21.2.5 *Initiation sequences*

The initiation sequences should give good fragmentation and displacement of the rock and, at the same time, maintain the vibration levels within an acceptable limit.

The size of the blastings should be as large as possible to avoid excessive movement of the machinery, road traffic interuptions, etc., and to do this, sequential blasting machines or milisecond delays are used.

Through cuts

The usual patterns are rectangular, Fig. 21.4, or in equilateral triangle, Figs 21.5 and 21.6.

The recommended triangular patterns are those found in Fig. 21.6, because the one shown in Fig. 21.5 gives an irregular slope profile.

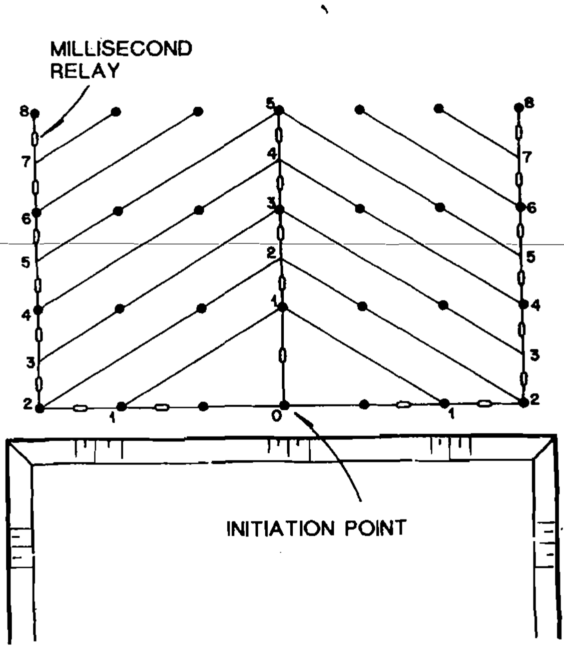


Fig. 21.4. Rectangular pattern with initiation sequence in V 1.

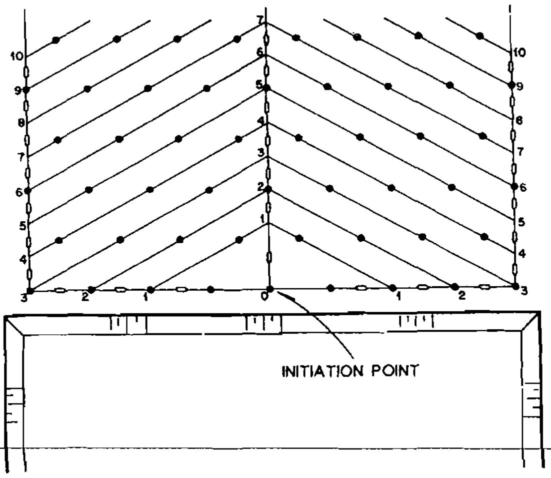


Fig. 21.5. Triangular pattern with initiation sequence in V1.

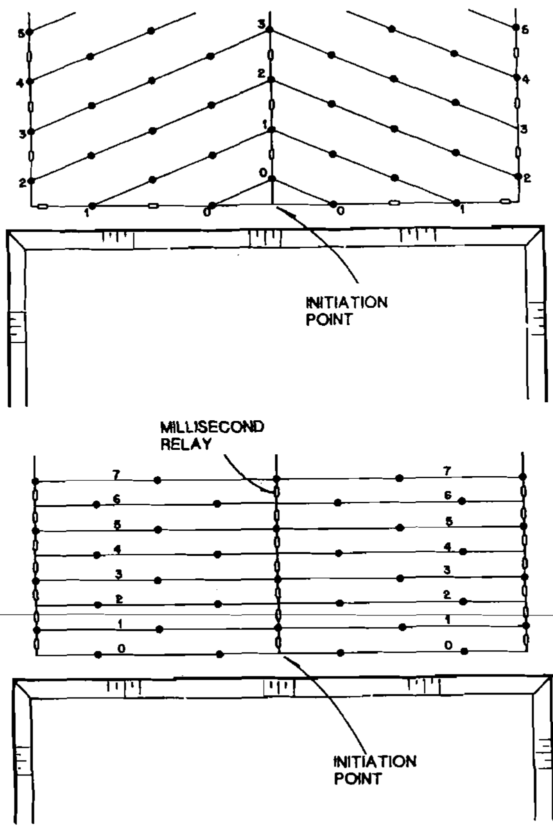
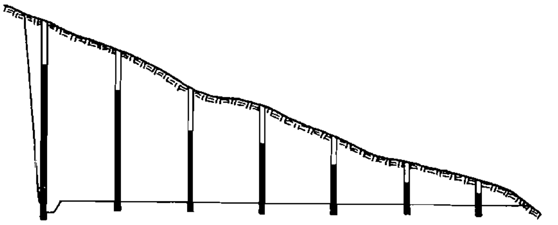


Fig. 21.6. Triangular patterns with initiation sequence in V and in line.

Side hill cut

The break direction of the blast can usually be normal to the direction of the outline or, even more frequently parallel to the same, Fig. 21.7. In the first instance, there is a risk of uncontrolled rocks rolling downhill and, subsequently, a more costly clean-up operation, as the broken rock will not be gathered in a pile.

When horizontal and vertical blastholes are combined in a blast, it is usually convenient to carry out the excavation in phases, taking away the broken rock from the first round before firing the next. If, because of operational necessities, blasting is one time, the initiation sequence should be as indicated in Fig. 21.8.



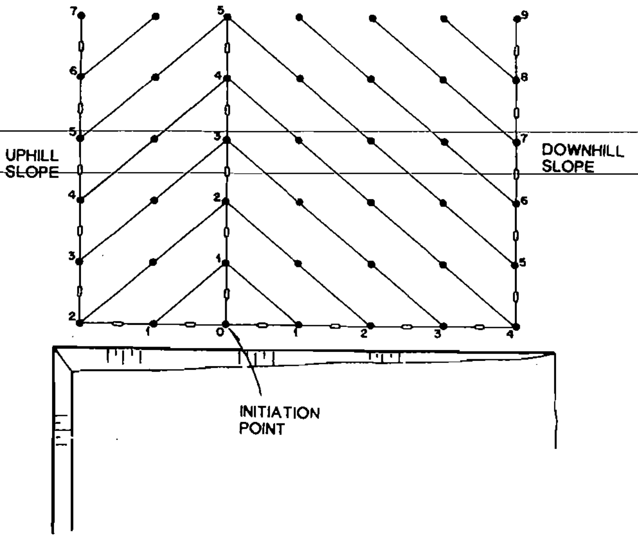


Fig. 21.7. Initiation sequence in V with break parallel to line design of highway.

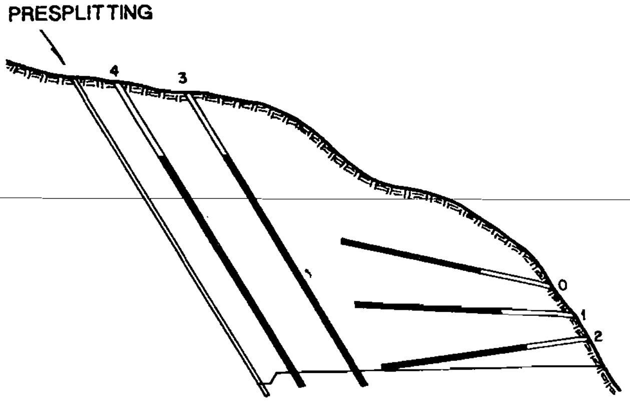


Fig. 21.8. Initiation secuence in section with vertical blastholes and lifters.

21.3 TRENCH BLASTING

Trench excavation with explosives presents a series of particular characteristics which call for a modification in design criteria for bench blastings and an adaptation of the same to different rock properties as well as taking special precautions such as control of vibration and fly-rock because these operations are usually carried out near urban areas.

Trenches are the lineal surface operations with a width of between 0.8 and 3 m, and a depth that can oscillate between 0.5 and 5 m.

They are used in drainage constructions, sewer systems, water and electrical conducts, gasoducts and oleoducts.

As the rock is more confined in these blastings, higher powder factors are used than in conventional bench blasting. This circumstance, together with more closed drilling patterns, results in a higher cost operation.

21.3.1 *Drilling diameters*

Blasthole drilling is always done with small diameters, using hand held hammerss in small urban operations, and light drilling rigs in larger excavations.

The calibers used depend upon the size of the trenches shown in Table 21.5, and the limitations imposed by the admissible vibration level.

Table 21.5.

Sizes of the trenches	Drilling diameters (mm)
Width – AZ < 1 Depth – H > 1.5	32 – 45
Width – AZ > 1 Depth – H > 1.5	50 – 65

Table 21.6.

Design parameter	Drilling diameter	
	< 50 mm	> 50 mm
Burden – B	26 D	24 D

Table 21.7.

Design parameter	Width Of The Trench – AZ		
	< 0.75 m	0.75 – 1.5 m	1.5 – 3 m
Number of rows	2	3	4
Spacing – S	AZ	AZ/2	AZ/2.6*

*In contour blastholes the spacing is reduced by 20%.

Photo 21.1. Trench blasting.

21.3.2 *Drilling patterns*

The drilling patterns depend basically upon the size of the excavation, with the burden value being set in function with the blasthole diameters, Table 21.6, and the spacing taken from the width of the excavation, Table 21.7.

21.3.3 *Subdrilling, stemming and inclination*

Subdrilling J is usually considered as 0.5 times burden value with a minimum value of 0.2 m.

The stemming is normally given a length equal to that of the burden.

The inclination of the drills usually favors bottom breakage, suggesting that the drilling should be carried out with angles with respect to the vertical between 26.5° and 18.5° (2:1 and 3:1).

21.3.4 *Charge distribution and types of explosives*

The most adequate explosives for trench excavation are those with high density and energy, as it is necessary to take maximum advantage of the total drilled hole volume. Therefore, gelatin dynamites, watergels and cartridged emulsions are the most common.

At the present time, two types of trench blastings are distinguished: conventional and smoothwall. The first have patterns in which the central blastholes are placed in front of the contour ones which break immediately afterwards, Fig. 21.9. The explosive charges are the same in all blastholes. The concentration of explosive in the column charge is considerably less than in the bottom charge (between 25 and 35%).

The type of blastings called smoothwall are characterized by having the central blastholes in line with the contour ones and use diferent explosive charges, depending upon the position of the blastholes, Fig. 21.10. In the central holes the bottom and column charges are increased and in the contour holes they are reduced, while the stemming decreases to a length of 10 D in the latter.

In order to calculate the lengths of the bottom charges, the following equations are used, Table 21.8, where H is the depth of the trench to be excavated in m.

The column charges can be made up of explosive agents (e.g. ANFO) with less strength and energy, or with cartridges similar to those in bottom but with a smaller diameter.

As to the advantages of one or the other type of blasting, in the conventional the charging is more simple because it is the same in all blastholes; however there is more overbreak as the contour blastholes have a smaller breakage angle and are more confined, Fig. 21.11.

The smoothwall blastings have the advantage of rectangular or symmetrical drilling patterns which are easier to replan and reduce the volume of overbreak in the walls of the trench. On the other hand, the explosive charges become complicated as they are different in the contour and lateral blastholes, and the vibration levels are superior as a consequence of the larger charges of the latter.

Table 21.8.

Conventional blastings	Smoothwall blastings
All the blastholes	Central blastholes
$0.4 + (H - 1)/5$	$1.3\,[0.4 + (H - 1)/5]$
	Contour blastholes
	$0.7\,[0.4 + (H - 1)/5]$

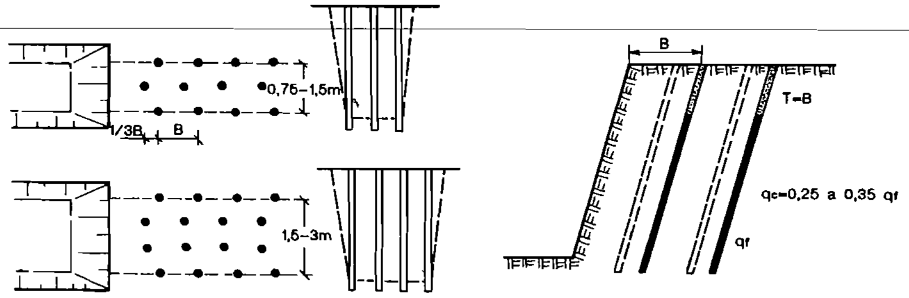


Fig. 21.9. Conventional trench blasting.

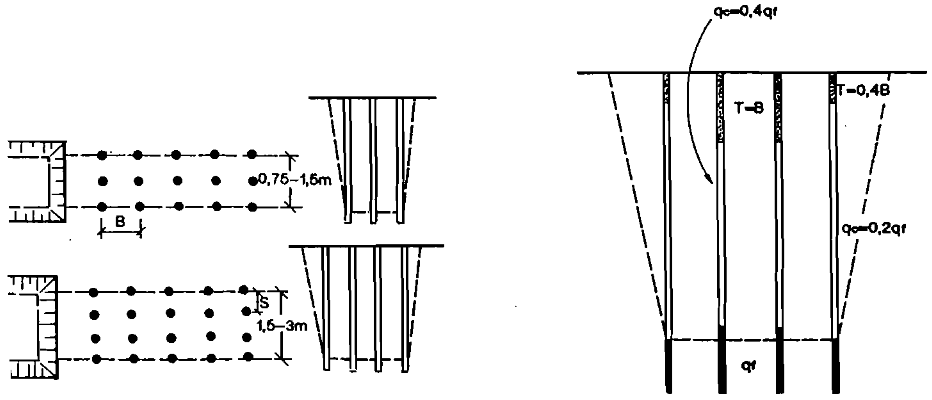


Fig. 21.10. Smoothwall trench blasting.

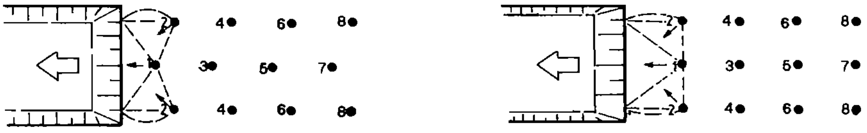


Fig. 21.11. Overbreak zones in the different types of blastings.

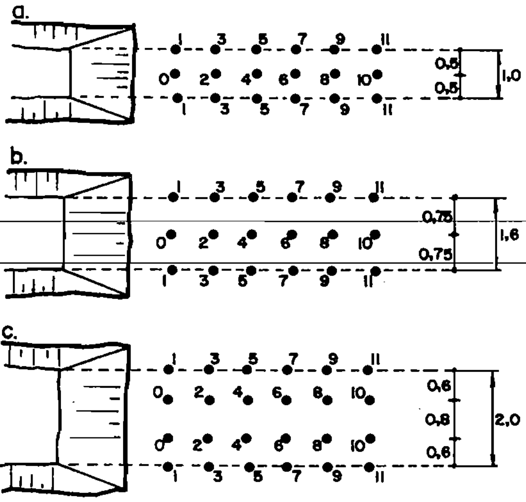


Fig. 21.12. Initiation sequences.

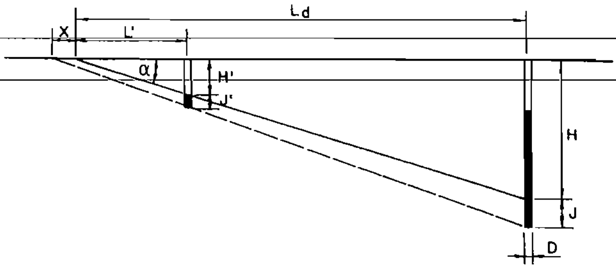


Fig. 21.13. Schematic diagram of a ramp cross section.

21.3.5 *Initiation sequences*

The charge initiation patterns in trench blastings should produce good fragmentation and rock breakage while, at the same time, maintaining overbreak at a minimum.

In Fig. 21.12 the recommended initiation sequences are indicated, depending upon the number of blasthole rows.

In order to increase the length of the rounds, sequenced blasting machines or milisecond delays can be used.

21.3.6 *Control of alterations*

Owing to the confinement and high powder factors used in this type of work, it is absolutely necessary, in inhabited areas, to use protection systems against flyrock.

As to vibration control, which is discussed in detail in Chapter 33, the most frequent measures taken are based upon the reduction of the operative charge by means of:

- Decreasing the drilling diameter.
- Using deck charges, and
- Dividing the blasts into phases.

21.4 RAMP BLASTING (SINKING CUT)

The continous deepening of the surface operations make ramp blastings obligatory every time a new level is opened.

According to the design in Fig. 21.13, for a ramp with a depth H, the projected length LD is:

$$\mathrm{LD} = \frac{H}{\mathrm{tag}\,\alpha} = \frac{H \times 100}{P_r}$$

where: α = Ramp inclination in degrees, P_r = Percentage of the ramp grade.

In the theory and design of blasts, there is a constant relationship between the size of the burden B and the subdrilling J, for a given diameter of blasthhole D, so:

$$B = K_1 \times J$$

On the other hand, it has been proved that for spherical charges of J length, burried under a stemming length H, that the craters have profiles that are close to the optimum if the relationship Charge Depth/Blasthole Radius is:

$$\frac{(H' + \tfrac{1}{2}J')}{\tfrac{1}{2}D} = 19 \pm 1$$

Taking that H is the cut depth in the most superficial line of the ramp, the following exists: that:

$$J' = 2.4\,D$$

and

$$H' = 8.3\,D$$

In Fig. 21.11, it is observed that because of triangle similarity:

$$\frac{H' + J'}{LD' + X} = \frac{H + J}{LD + X} = K$$

where: X = Horizontal distance between the theoretical line of the blasthole bottoms and the beginning of the ramp.

$$X = \frac{(H' + J') \times (H/\mathrm{tag}\,\alpha) - (H + J) \times (H'/\mathrm{tag}\,\alpha)}{(H + J) - (H' + J')}$$

The former equations can be applied to the calculations of the designs of each of the rows in a ramp blasting.

As the calculations are repetitive, it is common to use computers, as indicated in the adjoining listing, Table 21.9.

Nomographs are also used as the one in Fig. 21.14, made up for initial data that has already been established. In this case with $D = 223$ mm, $H = 12$ m, $J = 1.8$ m, $P_r = 8\%$.

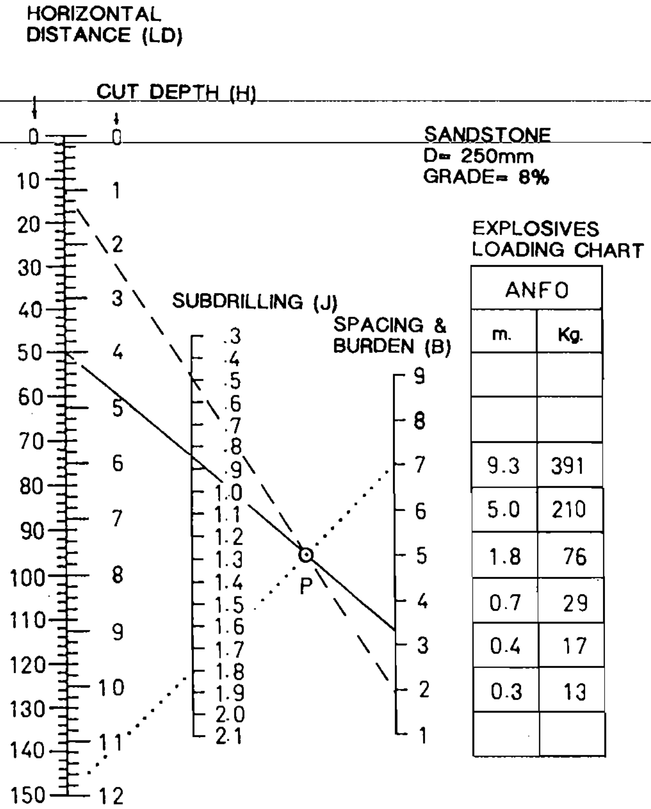


Fig. 21.14. Nomograph (Chung).

Table 21.9. Design for ramp blasting. Blasthole diameter (mm) = 250; Bench height (m) = 12; Design B = S (m) = 7; Subdrilling (m) = 61.8; Grade of ramp (%) = 8.

Row	Distance	Height	Burden	Subdrilling
1	143.00	11.44	6.73	1.73
2	136.26	10.90	6.48	1.66
3	129.78	10.38	6.23	1.60
4	123.54	9.88	6.00	1.54
5	117.53	9.40	5.77	1.48
6	111.75	8.94	5.56	1.43
7	106.19	8.49	5.35	1.37
8	100.84	8.06	5.15	1.32
9	95.69	7.65	4.95	1.27
10	90.73	7.25	4.77	1.22
11	85.96	6.87	4.59	1.18
12	81.37	6.50	4.41	1.13
13	76.95	6.15	4.25	1.09
14	72.70	5.81	4.09	1.05
15	68.60	5.48	3.93	1.01
16	64.67	5.17	3.79	0.97
17	60.88	4.87	3.64	0.93
18	57.23	4.57	3.51	0.90
19	53.72	4.29	3.37	0.86
20	50.34	4.02	3.25	0.83
21	47.09	3.76	3.12	0.80
22	43.96	3.51	3.01	0.77
23	40.95	3.27	2.89	0.74
24	38.05	3.04	2.78	0.71
25	35.26	2.82	2.68	0.69
26	32.58	2.60	2.58	0.66
27	29.99	2.39	2.48	0.63
28	27.51	2.20	2.39	0.61
29	25.11	2.00	2.30	0.59
30	22.81	1.82	2.21	0.56
31	20.60	1.64	2.13	0.54
32	18.46	1.47	2.05	0.52
33	16.41	1.31	1.97	0.50
34	14.44	1.15	1.90	0.48
35	12.53	1.00	1.82	0.47
36	10.71	0.85	1.76	0.45
37	8.94	0.71	1.69	0.43
38	7.25	0.58	1.63	0.41
39	5.62	0.44	1.56	0.40
40	4.05	0.32	1.51	0.38
41	2.54	0.20	1.45	0.37
42	1.09	0.08	1.39	0.35

Photo 21.2. Ramp blasting for the opening of a new bench.

In Fig. 21.15, the initiation sequence in a ramp using relays of detonating cord and initiating the blast in the deep-end to form a void that serves as a cut.

21.5 BLASTING FOR GROUND LEVELING

Ground leveling blasts are typical in preparing lots for buildings, industrial installations, etc. They should be carried out by experienced personnel because a precise control over the following is necessary:

– The fragmentation, because it must be loaded and hauled in equipment that can travel the city streets.

– The vibrations, airblast and projections must be kept under the threshold of safety because frequently other buildings and structures are nearby.

21.5.1 *Drilling diameters*

The blasthole diameters are in function with the bench height, which is usually small, and the maximum charge weight per hole depends upon the admissible level of vibrations.

It is possible to establish an initial approximation to the most adequate diameter from the bench height with the equation:

$$D = H/60$$

although usually it is found that the most common diameters are in the range of 38 to 65 mm.

21.5.2 *Drilling length*

As the heights of the cuts are small, the blastholes are drilled with inclinations close to 60°, because they give better fragmentation and swelling, a better take off from the floor and a lower vibration level.

The drilling length is calculated by the following equation:

$$L = \frac{H}{\cos\beta} + \left[1 - \frac{\beta}{100}\right] \times J$$

where: β = Blasthole angle with respect to the vertical (degrees), H = Mean height of the terrain that has to be broken by the blasthole (m), J = Subdrilling that depends upon the type of rock and the diameter of the drill, Table 21.10.

Table 21.10.

Design parameter	Uniaxial compressive strength (MPa)			
	Low < 70	Medium 70-120	High 120-180	Very high > 180
Subdrilling – J	10 D	11 D	12 D	12 D

Table 21.11.

Design parameter	Uniaxial compressive strength (MPa)			
	Low < 70	Medium 70–120	High 120–180	Very high > 180
Powder factor – CE (kg/m^3)	0.30	0.35	0.42	0.49
Stemming – T	35 D	34 D	32 D	30 D

21.5.3 *Charge and stemming distribution*

In Table 21.11, the powder factors and the stemming lengths are indicated for the different rock types.

Occasionally. the stemming can be reduced to give a larger volume of blasthole for the explosives, but never below 25 D to avoid airblast and flyrock.

As to the type of explosives, taking into consideration that the blastholes are short, high strength and high density explosives are used.

21.5.4 *Drilling design*

The burden is calculated from the following equation:

$$B = \left[\frac{Q_b}{\frac{S}{B} \times \frac{H}{\cos\beta} \times CE}\right]^{0.5}$$

where: Q_b = Explosive charge per blasthhole (kg), S/B = Ratio between Spacing and Burden. See Table 21.12, H = Mean height of the terrain (m), CE = Powder factor (kg/m^3), β = Angle with respect to the vertical (degrees).

The burden should be calculated for each row whenever the mean height of the terrain varies.

Special precautions should be taken in the marking out

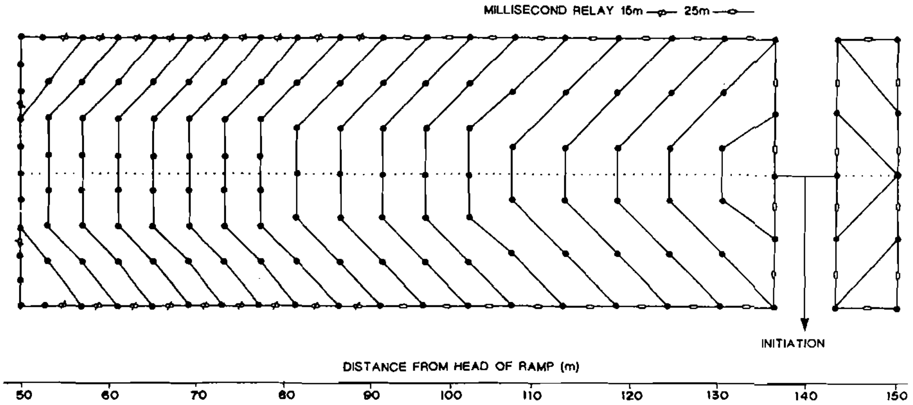


Fig. 21.15. Initiation sequence with milisecond relays in a ramp blasting.

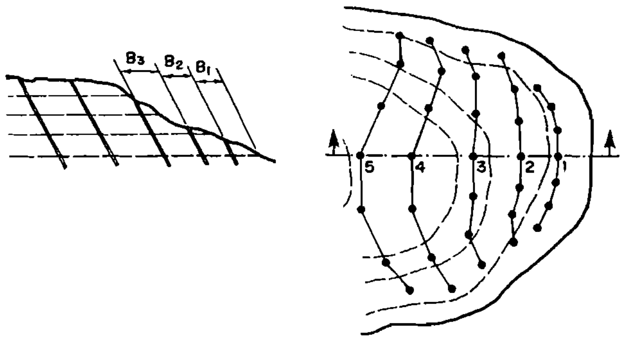


Fig. 21.16. Example of a leveling blasting.

of this type of blastings because of the difficulty that the irregular terrain imposes.

21.5.5 *Initiation sequences*

As the lateral blastholes usually shoot to a free exit, each of the rows can be initiated with the same delay timing or be sequenced in *V* if it is necessary to increase the time of the round because of vibration limitations.

21.5.6 *Blasts with horizontal blastholes*

In some projects, when the bench heights are too low and it is essential to leave the final cut suface as smooth as possible, horizontal blastholes provide numerous advantages, such as:

- Lower specific drilling and charge.
- Shorter time cycles.
- Less toe problem.
- Reduced risk of cut offs, improving safety.
- Less need for accessories.

The only disadvantages are:

- The broken rock must be removed before drilling the next module.
- The fragments are larger and the appearance of boulders is more probable.
- The charging of blastholes is more difficult.

The drilling diameters must be small to achieve good results. Therefore, supposing that the heights of the cuts are under 1.5 m, for 38 mm blasthole diameters the spacing between them is calculated, at first, with the following equation:

$$S = 0.7 \times H^{-1.1}$$

where: S = Spacing (m), H = Excavation height (m), under 1.5 m, Fig. 21.17.

Depending upon the capacity of the drilling equipment, the angle of the blastholes will be between 0° and 10°.

As to the charges, the blasts are fired to an ample free face, with a single row, and rarely need a bottom charge, only the column charge.

The stemmings should have a minimum length of 10 *D* to 20 *D*, according to the profile of the terrain and work conditions.

Table 21.12.

Design parameter	Uniaxial compressive strength (MPa)			
	Low <70	Medium 70-120	High 120-180	Very high >180
Ratio Spacing/ Burden - S/B	1.25	1.20	1.15	1.15

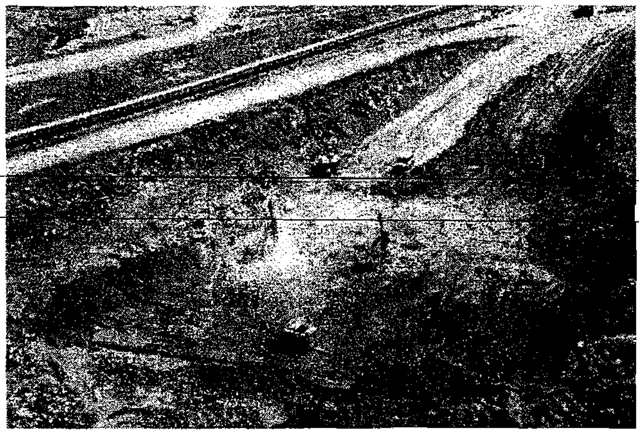


Photo 21.3. Drilling in a blast without a vertical free face.

The typical powder factors in levelling blasts with horizontal blastholes are around 0.2 kg/m^3.

21.6 BLASTINGS FOR FOUNDATIONS

Excavations with explosives for foundations in rock have the following problems:

- There is no free face, making fragmentation and swelling of the rock more difficult.
- The breakage of the rock should adapt itself to the profile established in the project without producing overbreak.
- There are limits for the vibrations and projections when the work is carried out in urban areas.

21.6.1 *Drilling diameters and lengths*

The drilling diameter and blasthole length are selected as explained in foundation blasting.

21.6.2 *Charge and stemming distribution*

The powder factors and the stemming lengths are decided from the compressive rock strength, Table 21.13.

If the stemming lengths are reduced, some additional safety precautions must be taken against flyrock.

The explosives used are generally of high strength and density.

21.6.3 *Drilling patterns*

Apply the equation and the relationship between spacing

Table 21.13.

Design parameter	Uniaxial compressive rock strength (MPa)			
	Soft	Medium	Hard	Very hard
	<70	70-120	120-180	>180
Powder Factor – CE (kg/m^3)	0.35	0.42	0.50	0.60
Stemming – T	35 D	34 D	32 D	30 D

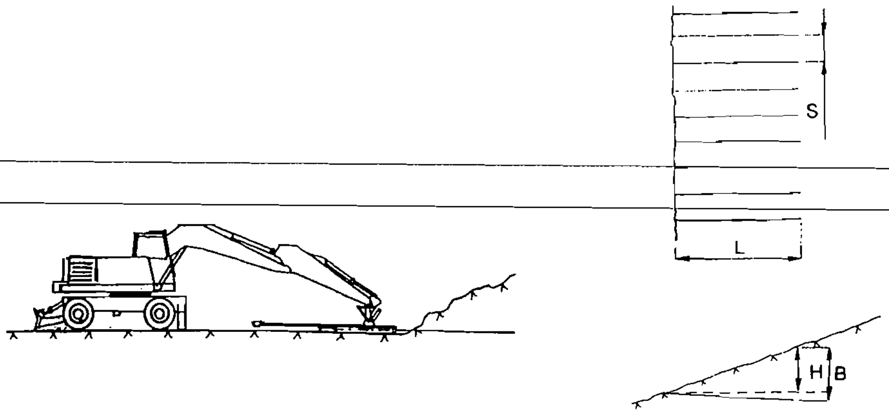


Fig. 21.17. Leveling blasts with horizontal blastholes.

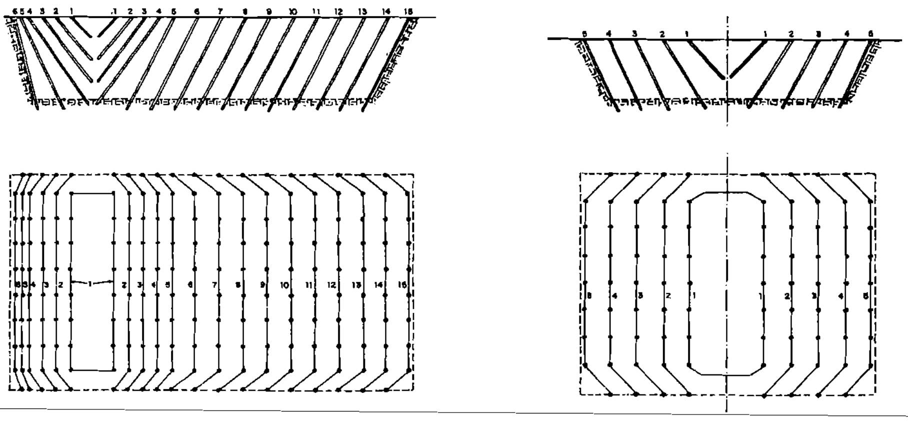


Fig. 21.18. Blasthole layout in long and narrow excavations.

and burden as suggested beforehand for foundation blasting.

As there is no free face, part of the blastholes should be angled in a wedge form to create the cut and the rest are drilled with the chosen inclination, which is usually around 60°, Fig. 21.18. In the cut blastholes, the charge concentration should be at least 30% more.

In some cases, the excavations are even carried out in phases, opening lateral ditches or trenches so as to form slopes which act as free faces in the bench blasts which are fired later.

21.6.4 *Initiation sequences*

The initiation sequences can be similar to those shown in Fig. 21.18, but they actually will be established from the vibrographic studies, as quite often there are limitations in the charge weight per hole and even in the delay timings.

21.7 MINI-HOLE BLASTING

In certain environments, owing to the proximity of buildings and other structures, which must be protected, large quantities of explosives cannot be used and drilling diameters must be reduced.

For some years now, a new type of high strength and density explosive is on the market which permits mini-hole blasting with hole diameters of 22 mm.

The explosive is a gelatin of nitroglycerin/nitroglycol and nitrocelulose to which PETN and amonium nitrate have been added. The principal characteristics are:

- Density . . . . 1.55g/cm^3
- Energy . . . . 5.5 MJ/kg
- Relative weight strength incomparison to ANFO 127
- Detonation velocity . . . . 6000 m/s
- Water resistance . . . . Excellent

The charges, of about 80 g, are prepared in small plastic wrappings of 17 × 275 mm with an inner space for the detonator.

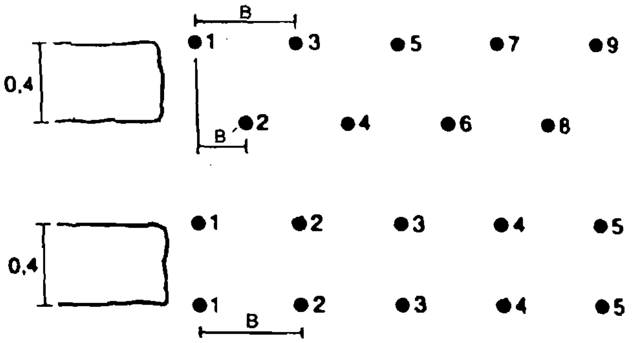


Fig. 21.19. Initiation sequence of mini-hole blastings in trenches for cables.

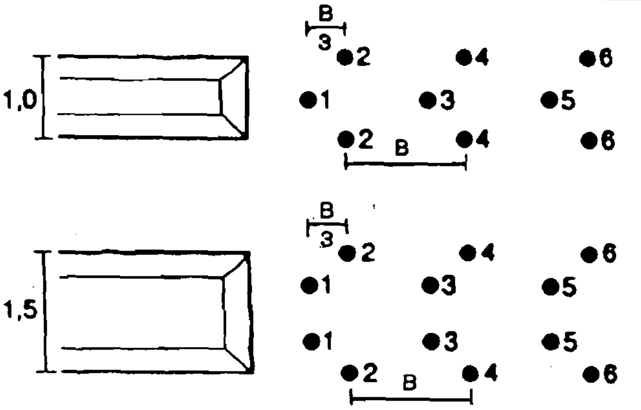


Fig. 21.20. Initiation sequences of mini-hole blastings in trenches for pipelines.

As a better use of the explosive energy has been achieved, the necessary quantities are about half that of a normal explosive, thereby reducing vibrations and flyrock.

Some of the most frequent applications for mini-hole blastings are found in the next paragraphs.

21.7.1 *Trenches for cables*

The trenches for cables usually have a limited depth which makes them especially suitable for mini-hole blasts, Fig. 21.19. It is recommended that the 22 mm diameter blastholes be drilled with inclinations of 18° (3:1), which means that the applicable rules for calculation in trenches with a depth between 0.3 and 0.6 are the following:

$$L = H + 0.2$$
$$B = 0.018\,D$$
$$T = B$$
$$Q_b = 0.2 \times H$$

where: H = Depth of trench (m), D = Diameter of blasthole (mm), B = Burden (m), T = Stemming (m), Q_b = Charge of high explosive per blasthole (kg).

21.7.2 *Trenches for pipelines*

If the depth of the trenches is under 0.8 m, the mini-hole blasting method is an adequate procedure as it permits a better use of the explosive energy, therefore reducing the risk of flyrock and lowering vibration levels, Fig. 21.20.

Maintaining the basic parameters indicated in trenches for cables, as the widths of the cuts are larger, the charges are calculated with the following equation:

$$Q_b = 0.03 + (H - 0.3) \times 0.25$$

where: H = Depth of the trench between 0.3 and 0.6 m.

The number of rows of blastholes is determined in function with the width of the trench, Table 21.14.

21.7.3 *Holes for posts and beams*

The use of explosives for the opening of holes in rock meant to hold posts or vertical beams has existed for years, but the conventional charges left crater-shaped holes which required different methods of support and also filling material.

When the holes have a diameter under 0.6 m, the mini-hole blasting technique, with special explosives, has opened a new field. The blastholes of 22 mm diameter are drilled parallel, leaving one or two of larger diameter empty, and charging the rest with special high strength explosive up to nearly 6 cm from the surface, Fig. 21.21.

The powder factors are high, as in the parallel hole cuts.

To achieve adequate fragmentation and expulsion of the fragmented rock, there should be at least 60 ms delay timing in the initiation sequence.

21.8 PREBLASTINGS

The equipment for direct breakage used in surface operations has limited applications imposed by the geomechanic characteristics of the massive rock. These can be expressed in terms of seismic velocity propagation as shown in Fig. 21.22.

Table 21.14.

Width of the trench	Number of rows of blastholes
0.8 – 1.2	3
1.5 – 2.0	4

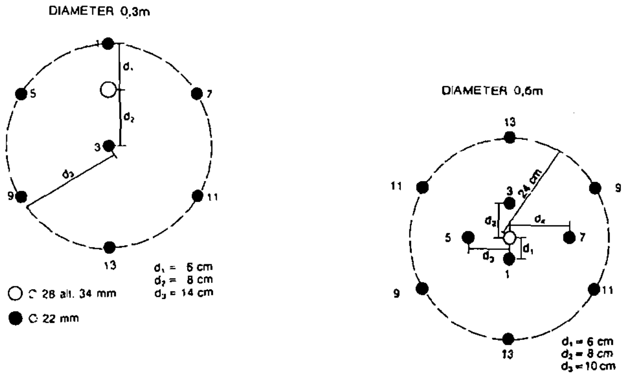


Fig. 21.21. Drilling patterns for 30 cm and 50 cm diameter holes.

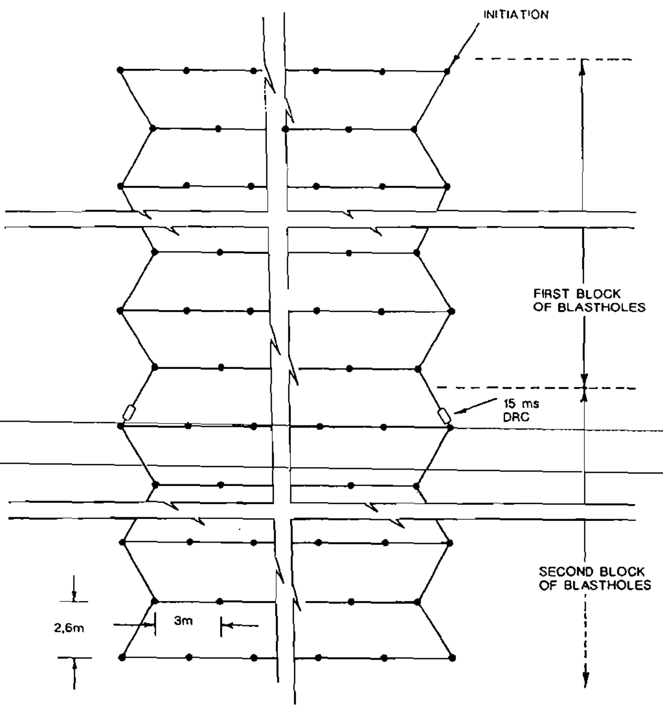


Fig. 21.22. Preblasting fired with detonating cord of low core load and milisecond relays.

Table 21.15.

Machinery	Bench height H(m)	Drilling diameter D (mm)
Crawler tractors	2-4	50-125
Hydraulic and rope shovels, draglines and bucket wheel excavators	10-15	125-250

Table 21.16.

Excavation equipment	Seismic velocity (m/s)	Powder factor (g. ANFO/m^3)
Crawler tractor		
(575 kW)	3000	230
(343 kW)	2500	130
(250 kW)	2000	130
(160 kW)	1200	80
Rope shovel	1800	180
Hydraulic shovel	2000	190
Dragline	1500	220
Bucket wheel excavator	1200	110

In some cases, although the excavation may be possible, the yield may be so low as to render the operation anti-economical in comparison to other alternative methods.

Preblasting is a technique that consists in increasing the natural fracturation of the massive rock with very little displacement, by using explosives. This improves the production rates of the operation with a relatively low cost.

The drilling diameters and bench heights depend basically upon the machines used after the preblasts, Table 21.15. In many cases, the height of the levels is also limited by the length of the drill steel because these operations should be one-time, eliminating unnecessary work.

The powder factors in each case depend upon the equipment for breakage and loading, and the compressive rock strength.

In Table 21.16, the approximate relationships between the maximum seismic velocities and the powder factors, such as ANFO, are indicated, to be able to properly carry out the preblastings. It is important to take into account that as the velocities are reduced, the explosive charges should do the same.

The drilling patterns may be square or staggered with $B = S$.

As to the blasthole inclination, with small diameters and distances, and the only free face which is usually the work platform itself, it is recommended that the angles be close to 45°, which requires a subdrilling between 15 and 20 *D*. In large diameters with higher benches, if the drilling rigs are rotary percussive, the drilling can be carried out with inclinations between 15° and 30°, and if the rigs are rotary, between 15° and the vertical.

The priming should be done with detonating cord, and if detonators are used, place them at the top.

The size of the blasts should be as large as possible in order to acheive maximum efficiency from the loading and haulage equipment.

In general, there should be no problem of flyrock if the stemming is done correctly with a minimum length of 35 *D*, and the only limitation could be the vibrations. It should not be forgotten that in this type of blasts the charges are more confined and, in comparison with a conventional round, the vibrations are greater.

To reduce this problem, it is possible to use milisecond relays as indicated in Fig. 21.22, but with a timing that is not too long as the main direction of rock movement is towards the surface and there is a certain risk of cut-offs.

REFERENCES

Chung, C.S.: *Computerized sinking cut design in open pit mining.* 14th Canadian Rock Mechanics Symposium, Vancouver, 1982.

Gustaffson, R.: *Swedish Blasting Technique.* SPI, 1973.

Langefors, U. & B. Kihlstrom: *Voladura de Rocas.* Edit. URMO, 1973.

Hagan, T.N.: *Initiation sequence, delay timings and cut-offs.* AMI, 1982.

Lopez Jimeno, E. & C. Lopez Jimeno: *Las Voladuras en Cráter y su Aplicación a la Minería.* VII Congreso Internacional de Minería y Metalurgia. 1984

Musil, P.: *Effect of blasting on output increase of bucket wheel excavators.* Bulk Solids Handling, December, 1981.

Petersson, S-A.: *A shorter guide to trench blasting.* Nitio Nobel AB, 1985

Tamrock: *Handbook on Surface Drilling and Blasting.* 1983.

CHAPTER 22

Blasting for tunnels and drifts

22.1 INTRODUCTION

The use of the underground, for mining as well as for civil engineering projects, requires the driving of drifts and tunnels in a larger number each day.

During the past few years, mechanical excavation with drifting and tunnelling rigs has advanced considerably, excavating rock with a compressive strength of up to 250 MPa. In hard rock, the tunnelling rigs have a larger field of application and offer several advantages such as: perforation without harming the surrounding rock, a smooth cut surface that reduces the necessity for support and/or a concrete lining, less personnel, etc.

However, excavation with explosives is still widely accepted as the aforementioned method has its inconveniences:

- Rigid work system as the sections must be circular.
- The ground to be drilled must not have important variations or geological upsets.
- The curves should have a radius over 300 m.
- The initial excavation is costly, and
- Personnel must be highly specialized.

Fragmentation with drilling and blasting solves most of these problems because although the sections are large, the excavations can be carried out in stages with advance drifts, lateral breakage and/or benching, plus the fact that modern jumbos can cover large sections with varying shapes. The remaining rock can be left in good condition with contour blasting using smooth blasting and presplitting techniques which are better adapted to the lithological changes of the ground, and require less machine time, which means the equipment can be sent on to other jobs.

The basic drilling cycle is composed of the following operations:

- Blasthole drilling.
- Charging.
- Blasting.
- Ventilation.
- Scaling and grouting (if necessary).
- Loading and haulage, and
- Setting up of the new round.

In the following paragraphs the present day techniques for tunnelling and drifting, as well as the calculations for drilling patterns and charges are revised.

22.2 ADVANCE SYSTEMS

The manner or pattern used to attack the tunnel and drift sections depends upon various factors:

- Drilling equipment used.
- Time available for the operation.
- Type of rock.
- Type of support, and
- Ventilation systems.

In good rock, the tunnels with sections under 100 m^2 can be excavated by the full face method. Larger tunnels which cannot be covered by the drilling unit, or those where the geomechanic properties of the rock do not permit the full face method, have to be excavated by the top heading method, Fig. 22.1.

The procedure consists of dividing the tunnel in two parts, an upper or top heading, and a lower in bench or stoping holes. The top heading is excavated as a drift and the stoping holes, which take longer than the top heading advance, are carried out by benching.

Benching can be vertical, in which case it will be necessary to have a crawler drill with a feed that is not too large in order to avoid positioning problems near the side walls. The advantage of this system is that the whole bench can be drilled and blasted without interruption and simultaneously with the top heading, Fig. 22.2.

Horizontal benching uses the same drilling equipment as for the top heading and also the same procedure for charging the explosive and mucking, but it has the problem of work interruptions.

When the rock is of poor quality, the tunnel is usually divided into several smaller sections. A common technique is to open a pilot heading in the top heading with one or two lateral stoping holes. The pilot heading, which is mainly for inspection, is done ahead of the stoping and may even be opened before starting the lateral drilling which provides better work ventilation. The excavation of the top heading is usually completed before starting rock breakage of the lower section, although in wide tunnels it may be carried out simultaneously by establishing an access between the tunnel floor and the top heading by means of a lateral ramp.

At present, one of the tunnelling procedures most widely used is known as the *New Austrian Tunneling Method.* It roughly consists of excavation in stages, such as indicated. After opening the advance drift in the upper half section, the lateral stoping holes can be carried out either simultaneously or by dephasing, using the drift cavity as a free face and placing the contour blastholes so as to achieve a permanent profile with as little damage possible to the rock, using the smooth wall blasting

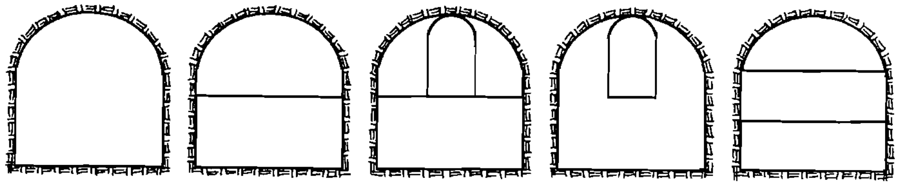


Fig. 21.1. Systems for advance in tunnel and drift excavations.

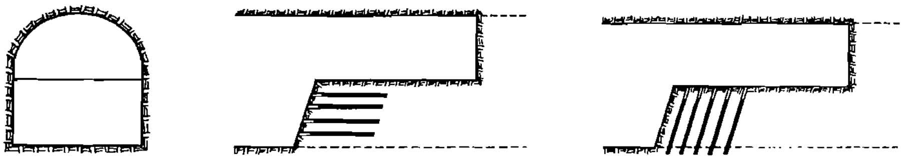


Fig. 22.2. Horizontal and vertical benching in a tunnel with the advance in two sections.

technique. Later, after mucking, shotcrete lining is built in the area to avoid decompressions and loss of compressive strength qualities of the rock.

At a certain distance from the face, which is usually the same as the advance of the rounds, the final lining should be built in by any of the existing procedures.

The excavation of the lower section is also carried out in stages, in the center part by benching and in the lateral masses with side stoping or smooth wall blasts. The drilling can be vertical or horizontal and the advance of the side stoping either simultaneous or dephased.

22.3 BLASTING PATTERNS FOR TUNNELS

The blasts in tunnels and drifts are characterized by the initial lack of an available free surface towards which breakage can occur; only the tunnel heading itself. The principle behind tunnel blasting is to create an opening by means of a cut and then stoping is carried towards the opening. The opening usually has a surface of 1 to 2 m^2, although with large drilling diameters it can reach up to 4 m^2. In fan cuts, the cut and *cut spreader* blastholes usually occupy most of the section.

Stoping can be geometrically compared to bench blasting although it requires powder factors that are 4 to 10 times higher. This is due to drilling errors, the demand made by swelling, the absence of hole inclination, the lack of co-operation between adjacent charges and, in some areas, there is a negative action of gravity as happens in lifter holes, Fig. 22.3.

Contour holes are those which establish the final shape of the tunnel and are placed with little spacing and directed towards the interior of the mass to make room for the drills in collaring and advance, Fig.22.4.

The position of the cut has influence on rock projection, fragmentation and also on the number of blastholes. Of the three positions, corner, lower center and upper center, the latter is usually chosen as it avoids the free fall of the material, the profile of the broken rock is more extended, less compact and better fragmented.

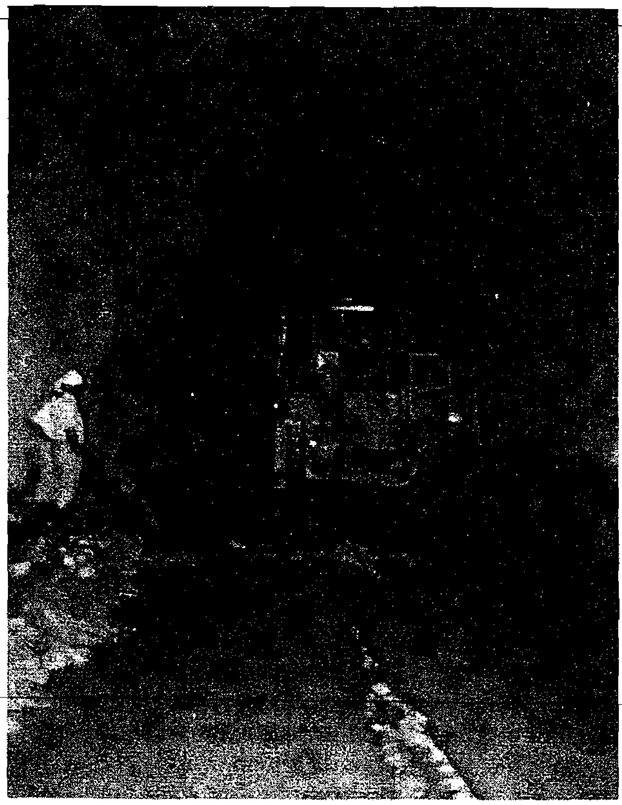


Photo 22.1. Pilot excavation and lateral stoping holes of the upper section of the pressure tunnel with a diameter of 12 m, in the Saucelle Power Station.

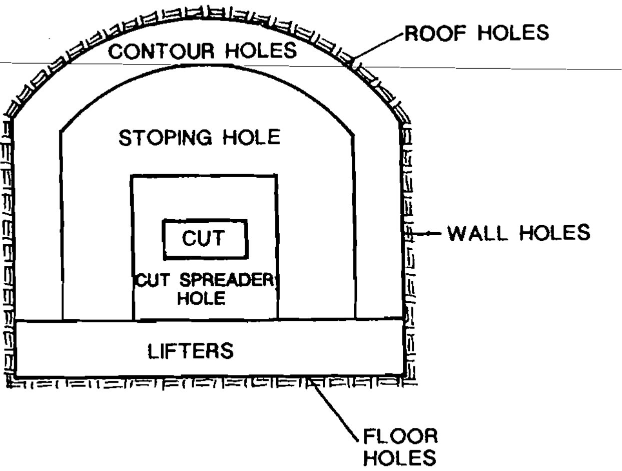


Fig. 22.3. Zones in tunnel blasting.

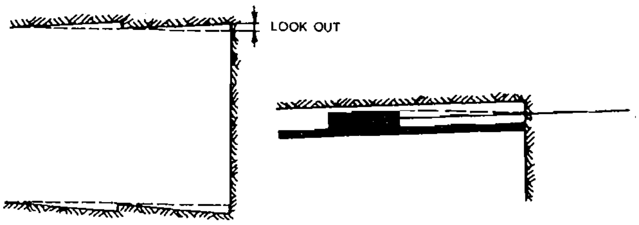


Fig. 22.4. Orientation of the contour blastholes to maintain the tunnel profile.

22.4 TYPES OF CUTS AND CALCULATION OF THE BLASTS

The blasts in tunnels and drifts are much more complex than bench blastings owing to the fact that the only free surface is the tunnel heading. The powder factors are elevated and the charges are highly confined. On the other hand, burdens are small, which requires sufficiently insensitive explosives to avoid sympathetic detonation and at the same have a high enough detonation velocity, above 3000 m/s, to prevent channel effects in the cartridge explosives placed in large diameter blastholes. This phenomena consists of the explosion gases pushing the air that exists between the column charge and the wall of the blasthole, compressing the cartridges in front of the shock wave, destroying the hot spots or excessively increasing the density of the explosive.

As to drilling, this has become more mechanized in the last decades, based upon the development of hydraulic jumbos, with one or various booms, automatized and more versatil. Because of this, the inclination has been towards parallel hole cuts as they are easier to drill, do not require a change in the feed angle and the advances are not as conditioned by the width of the tunnels, as happens with angled cuts.

Therefore, cuts can be classified in two large groups:

- Parallel hole cuts, and
- Angled hole cuts.

The first group is most used in operations with mechanized drilling, whereas those of the second have fallen in disuse due to the difficulty in drilling. They are only applied in small excavations.

In the following, the different types of cuts are explained in their order of importance, as well as calculation of the patterns and charges in the rest of the sections which are, generally speaking, independent from the type of cut applied.

22.4.1 *Cylindrical cuts*

At the moment, this type of cut is most frequently used in tunnelling and drifting, regardless of their dimensions. It is considered to be an evolution or perfection of the burn cuts which will be discussed later on. This type of cut consists of one or two uncharged or relief blastholes towards which the charged holes break at intervals. The large diameter blastholes (65 to 175 mm) are drilled with reamer bits which are adapted to the same drill steel which is used to drill the rest of the holes.

All the blastholes in the cut are placed with little spacing, in line and parallel, which explains the frequent use of jumbos which come with automatic parallelism.

The type of cylindrical cut most used is the four section, as it is the easiest one to mark out and execute. The calculation method for patterns and charges of this cut and for the rest of the tunnel zones, uses the Swedish theories recently up dated by Holmberg (1982), and simplified by Olofsson (1990), which will be studied below. Finally, other types of cylindrical cuts have been used with success and have been well experimented.

Advance per round

The advance of the rounds is limited by the diameter of the relief hole and the deviation of the charged holes. As long as the latter is maintained under 2%, the average advances X can reach 95% of the blasthole depth L.

$$X = 0.95\,L$$

In the four section cuts, the depth of the blastholes can be estimated by the following equation:

$$L = 0.15 + 34.1\,D_2 - 39.4\,D_2^2$$

where: D_2 = Diameter of empty hole (m).

When cuts of NB empty holes are used instead of only one large diameter drillhole, the former equation is still valid making

$$D_2 = D'_2 \times \sqrt{NB}$$

where D'_2 is the diameter of the empty blastholes.

Cut and 'cut spreader'

The general geometric pattern of a four section cut with parallel blastholes is shown in Fig. 22.5.

The distance between the central blasthole and those of the first section should not be more than 1.7 D_2 to obtain fragmentation and a satisfactory movement of the rock (Langefors and Kilhström, 1963). The conditions of fragmentation vary greatly depending upon the type of

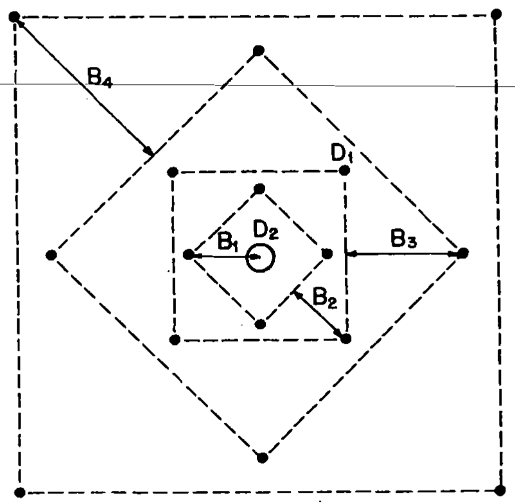


Fig. 22.5. Four section cut.

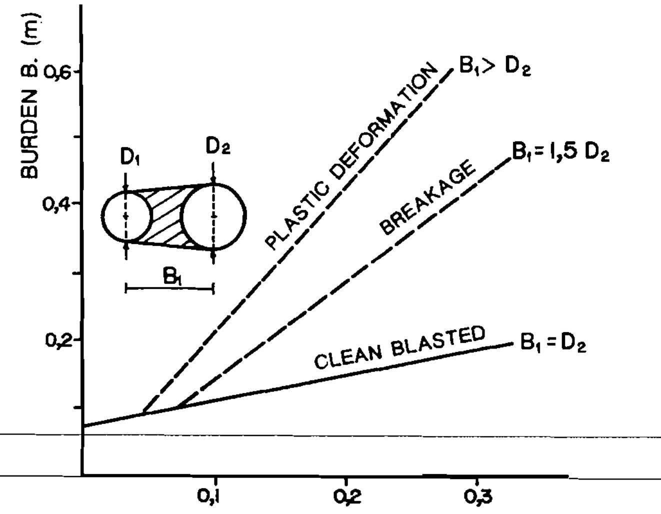


Fig. 22.6. Results of the blastings for different distances between charged and empty blastholes, and their diameters (Holmberg).

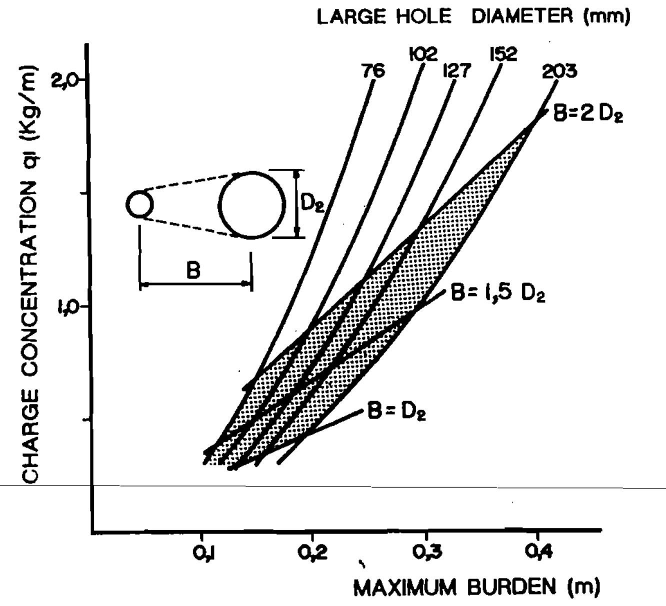


Fig. 22.7. Relationship between the lineal concentration of maximum burden and charge for the different diameters of relief blastholes (Larsson and Clark).

explosive, rock properties and the distance between the charged blasthole and the relief hole.

As reflected in Fig. 22.6, for burdens larger than 2 D_2, the break angle is too small and a plastic deformation of the rock between the two blastholes is produced. Even if the burden is under D_2, but the charge concentration is high, a sinterization of the fragmented rock and cut failure will occur. For this reason, it is recommended that the burdens be calculated from $B_1 = 1.5\,D_2$.

When drilling deviation is more than 1%, the practical burden is calculated from:

$$B_1 = 1.7\,D_2 - E_p = 1.7\,D_2 - (\alpha \times L + e')$$

where: E_p = Drilling error (m), α = Angular deviation (m/m), L = Blasthole depth (m), e' = Collaring error (m).

What usually happens is that the drilling is sufficiently good and the work is carried out with a burden value equal to one and one half times the expansion diameter. The lineal charge concentration is calculated from the following equation:

$$q_l = 55\,D_l \left[\frac{B}{D_2}\right]^{1.5} \times \left[B - \frac{D_2}{2}\right] \times \left[\frac{c}{0.4}\right] \times \frac{1}{\text{PRP}_{\text{ANFO}}}$$

where: q_l = Lineal charge concentration (kg/m), D_l = Drilling diameter (m), D_2 = Diameter of the relief blasthole (m), B = Maximum distance between holes and burden (m), c = Rock constant, PRP_{ANFO} = Relative weight strength of the explosive with respect to ANFO.

Frequently, the possible values of the lineal charge concentrations are quite limited as there is not an ample variety of cartridged explosives. This means that for a pre-fixed lineal concentration, the burden size can be determined from the former equation, although the calculation is a bit more complex, Fig. 27.7.

To calculate the rest of the sections, it is considered that some rectanglar openings of A_h width already exist and that the lineal charge concentrations q_l are known. The burden value will be calculated from:

$$B = 8.8 \times 10^{-2} \sqrt{\frac{A_h \times q_l \times \text{PRP}_{\text{ANFO}}}{D_l \times c}}$$

When there is a drilling error, such as that seen in Fig. 22.9, the free surface A_h differs from the hole distance A_h in the first section, for which

$$A_h = \sqrt{2}\,(B_1 - E_p)$$

and, by substituting this value in the former equation, the following occurs:

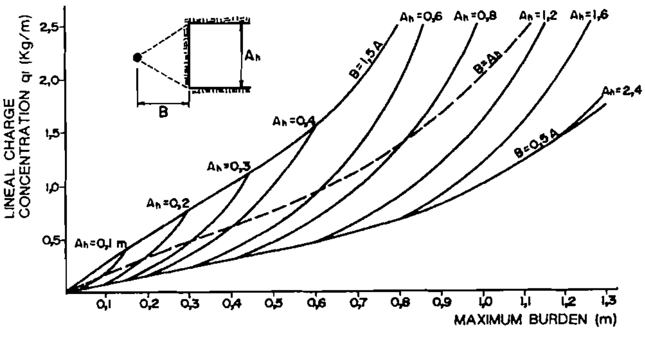


Fig. 22.8. Relationship between the lineal concentration of the charge and the maximum burden for different widths of the opening (Larsson and Clark).

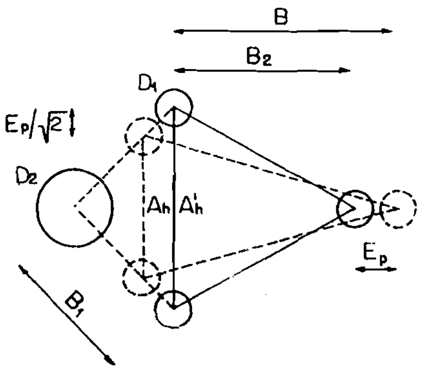


Fig. 22.9. Influence of blasthole deviation (Holmberg).

$$B = 10.5 \times 10^{-2} \sqrt{\frac{(B_1 - E_p) \times q_l \times \text{PRP}_{\text{ANFO}}}{D_1 \times c}}$$

This value must be reduced by the blasthole deviation to obtain the practical burden

$$B_2 = B - E_p$$

There are a few restrictions put on B_2, as it must satisfy:

$$B_2 \leq 2 A_h$$

if plastic deformation is to be avoided. If this is not true, the lineal charge concentration should be modified using the following equation:

$$q_l = \frac{540 D_1 \times c \times A_h}{\text{PRP}_{\text{ANFO}}}$$

If the restriction for plastic deformation is not satisfactory, it is usually better to choose a lower weight strength explosive in order to optimize fragmentation.

The aperture angle should also be less than 1.6 rad (90°). If not, the cut will lose its character of a four section cut. This means that:

$$B_2 > 0.5 A_h$$

Gustafsson (1973) suggests that the burden for each section be calculated with $B_2 = 0.7 B'$.

A rule of thumb to determine the number of sections is that the side length of the last section B should not be less than the square root of the advance. The calculation method for the rest of the sections is the same as for the second section.

The stemming lengths are estimated with:

$$T = 10 D_1$$

Some of the problems that can arise in blastings with parallel blasthole cuts are sympathetic detonation and dynamic pressure desensitization. The first phenomenon can appear in a hole that is adjacent to the detonating hole when the explosive used has a high degree of sensitivity such as all those with nitroglycerine in their composition. On the other hand, the dynamic pressure desensitization takes place in many explosives, and especially in ANFO because the shock wave of a charge can elevate the density of the adjacent charge above the critical or death density.

Desensitization problems can be attenuated by correctly designing the initiation sequences, sufficiently delaying the successive detonation of each blasthole so that the shock wave from the last shot disappears, allowing the explosive to recuperate its normal density and degree of sensitivity.

Hagan suggests that, in order to diminish these problems, the burn-cuts be carried out by placing three relief holes in such a manner that they act as a shield between the charged holes, Fig. 22.10

Hagan has also been able to prove that fine grade rock is more subject to cut failures than coarse grade, due to the larger volume of relief opening that is needed for the expulsion of the material.

As in burn-cuts each successive detonation enlargens the available space for expansion of the blastholes which have not yet fired, the burden can get increasingly larger, therefore placing the charges in a spiral Fig. 22.11.

a) *Double spiral burn-cut*

A central blasthole is drilled with a diameter between 75 and 200 mm, surrounded by smaller blastholes that are placed and charged in a spiral, Fig. 22.12.

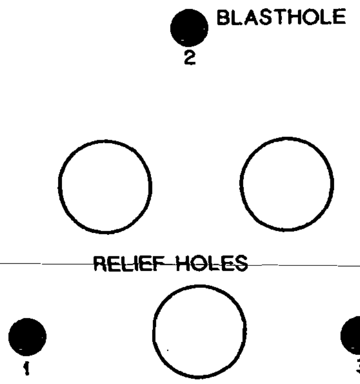


Fig. 22.10. Modified burn-cut to eliminate sympathetic detonation and dynamic pressure desensitization (Hagan).

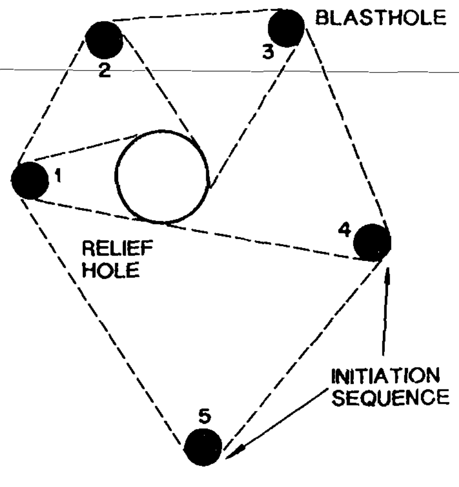


Fig. 22.11. Spiral burn-cut.

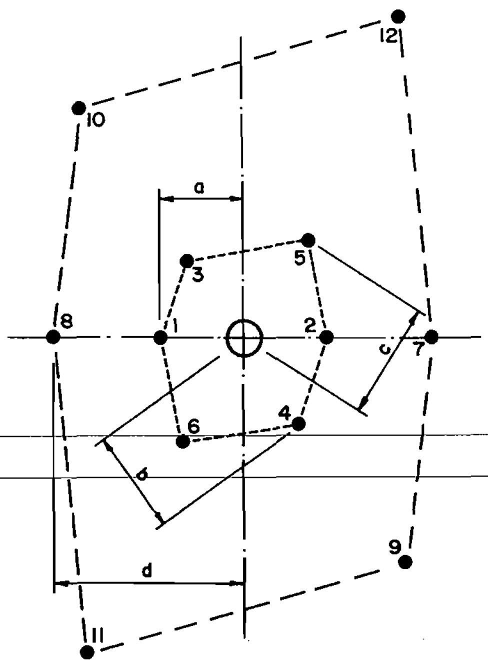


Fig. 22.12. Double spiral cut.

The blastholes 1-2, 3-4, and 5-6 are corresponding in each of their respective spirals.

b) *Coromant cut*
Consists in drilling two secant blastholes of equal diameter (57 mm), which constitute the free opening, in slit shape, for the first charges. A special drilling template is used to bore the first two holes as well as those of the rest of the cut, Fig. 22.13.

c) *Fagersta cut*
A central blasthole is drilled with a diameter of 64 or 76 mm and the rest of the charged blastholes, which are smaller, are placed according to Fig. 22.14.

This is a type of cut that is a cross between the four section cut and the double spiral, and is adequate for small drifts that use manual drilling.

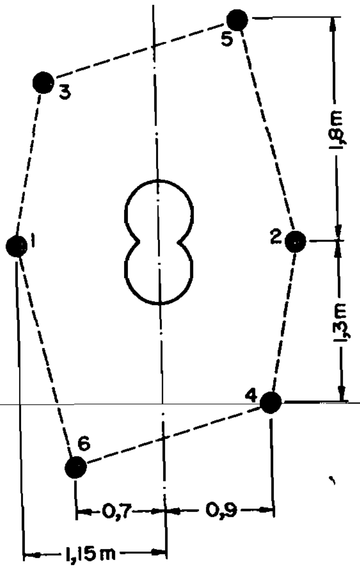


Fig. 22.13. Coromant cut.

Lifters
The burden for lifter holes placed in a row is calculated, basically, with the same equation that is used in bench blastings, taking into consideration that the height of the latter is equal to the advance of the round:

$$B = 0.9\sqrt{\frac{q_1 \times \mathrm{PRP}_{\mathrm{ANFO}}}{\bar{c} \times f(S/B)}}$$

where: f = Fixation factor. Generally 1.45 is taken to consider the gravitational effect and the delay timing between blastholes, S/B = Relationship between spacing and burden. It is usually considered equal to 1, c = Corrected rock constant.

$$\bar{c} = c + 0.05 \text{ for } B \geq 1.4\text{ m}$$
$$\bar{c} = c + 0.07/B \text{ for } B < 1.4\text{ m}$$

In lifters it is necessary to consider the lookout angle γ or inclination necessary to give a large enough space for the drilling rig to carry out the collaring for the next round. For an advance of 3 m, an angle of 3°, which has an equivalent of 6 cm/m, is enough, however it will logically depend upon the characteristics of the equipment, Fig. 22.15.

The number of blastholes will be given by

$$\mathrm{NB} = \text{Integer of}\left[\frac{AT + 2L \times \sin\gamma}{B} + 2\right]$$

where: AT = Tunnel width (m).

The practical spacing for the corner blastholes will be:

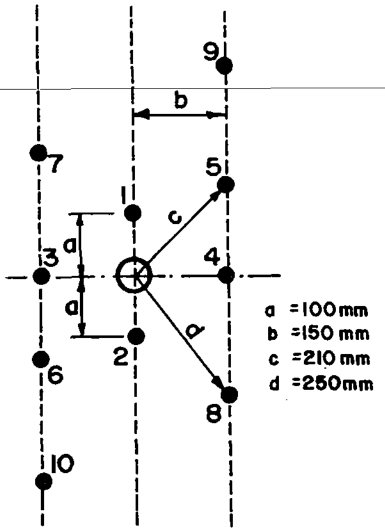


Fig. 22.14. Fagersta cut.

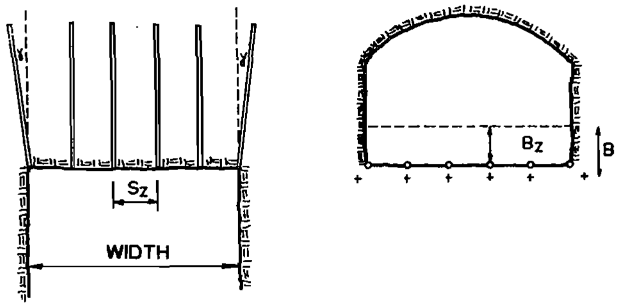


Fig. 22.15. Geometry of the lifters.

$$S'_z = S_z - L \sin \gamma$$

The practical burden B_z is obtained from

$$B_z = B - L \sin - E_p$$

The lengths of the bottom charges l_f and the column charges should be

$$l_f = 1.25\,B_z$$
$$l_c = L - l_f - 10\,D_1$$

The concentration of the column charge can be reduced to 70% of the bottom charge. However, the same concentration is usually used because of preparation time. The stemming is fixed in $T = 10\ D_1$ and the burden should comply with the following condition: $B \leq 0.6\,L$.

Stoping

The method for calculating the stoping holes is similar to the one used for lifters, only applying different Fixation Factor and Spacing/Burden relationship values, Table 22.1.

The concentration of the column charge for both types of blastholes should be equal to 50% that of the bottom charge.

Contour blasts

If the blast does not need contour or smooth blasting, the patterns are calculated as for lifters with the following values: Fixation factor, $f = 1.2$; Relationship S/B, $S/B = 1.25$; Column charge concentration, $q_c = 0.5\ q_f$, where q_f is the bottom charge concentration.

If contour blasting is to be carried out, the spacing between blastholes is calculated from:

$$S_c = KD_1$$

where K varies between 15 and 16. The ratio S/B should be 0.8.

The lineal charge concentration is determined in function with the drilling diameter. With blastholes of a caliber lower than 155 mm, the following equation is used:

$$q_{lc} = 90 \times D_1^2$$

where: D_1 is expressed in m.

Example of application

If a mine drift is to be excavated in rock ($c = 0.4$) by means of blasts with parallel blastholes and four section cut, knowing that the geometric dimensions and drilling data are:

- Tunnel width $AT = 4.5$ m,
- Abutment height 4.0 m,
- Height of arch 0.5 m,
- Relief hole diameter $D_1 = 102$ m,
- Drilling diameter $D_1 = 45$ mm,

Table 22.1.

Breaking direction of the stoping holes	Fixation factor f	S/B relationship
Upwards and horizontally	1.45	1.25
Downwards	1.20	1.25

- Lookout angle of the contour blastholes $\gamma = 3°$,
- Angular deviation $\alpha = 10$ mm/m,
- Collaring error $e' = 20$ mm.

The explosive to be used has a Relative Weight Strength with respect to ANFO of 1.09 (109%) and the available cartridges have a diameter of 25, 32 and 38 mm, which give lineal charge concentrations for a density of 1.2 g/cm^3, of 0.59, 0.97, and 1.36 kg/m respectively.

a) *Advance*

$L = 3.2$ m and $X = 3.0$ m.

b) *Cut*

First section

$B = 1.7 \times D_2 = 0.17$ m
$B_1 = 0.12$ m
$q_1 = 0.58$ kg/m $\rightarrow 0.59$ kg/m, with $d = 15$ mm
$T = 10 \times D_1 = 0.45$ m
$A_h = \sqrt{2} \times B_1 = 0.17$ m
Charge per blasthole $Q_b = 1.59$ kg.

Second section

$A_h = \sqrt{2}\,(0.12 - 0.05) = 0.10$ m
For $d = 25$ mm $B = 0.17$ m; $d = 32$ mm $B = 0.21$ m; $d = 38$ mm $B = 0.25$ m
As $B_2 \leq 2\,A_h$, cartridges of 32 mm are chosen.
$B_2 = 0.16$ m
$T = 0.45$ m
$A_h = \sqrt{2}\,(0.16 + 0.17/2) = 0.35$ m
$Q_b = 2.62$ kg.

Third section

$A_h = \sqrt{2}\,(0.16 + 0.17/2 - 0.05) = 0.28$ m
For larger diameter cartridges $q_l = 1.36$ kg/m
$B = 0.42$ m
$B_3 = 0.37$ m
$T = 0.45$ m
$A'_h = \sqrt{2}\,(0.37 + 0.35/2) = 0.77$ m
$Q_b = 3.67$ kg

Fourth section

$A_h = \sqrt{2}\,(0.37 + 0.35/2 - 0.05) = 0.70$ m
$B = 0.67$ m
$B_4 = 0.62$ m
$T = 0.45$ m
$A'_h = \sqrt{2}(0.62 + 0.77/2) = 1.42$ m, which is comparable to the square root of the advance, which means that no more sections are needed.
$Q_b = 3.67$ kg.

c) *Lifters*

With $d = 38$ mm $q_l = 1.36$ kg/m
$B = 1.36$ m
$NB = 5$ blastholes
$S_z = 1.21$ m
$S'_z = 1.04$ m
$B_z = 1.14$ m
$l_f = 1.43$ m
$l_c = 1.32$ m

$q_c = 0.7 \times 1.36 = 0.95$ kg/m $\rightarrow$ 0.97 kg/m with d= 32 mm
$Q_b = 3.20$ kg.

d) *Roof contour blastholes*

Cartridges of 25 mm with $q_l = 0.59$ kg/m are used.

$S_{ct} = 15 \times D_l = 0.68$ m
$B_{ct} = S_{ct}/0.8 - L \times \sin 3° - 0.05 = 0.62$ m
$q_{lc} = 90 \times D_l^2 = 0.18$ kg/m, which is considerably less than 0.59 kg/m
NB = $|\ 4.7/0.68 + 2\ |$ = 8
$Q_{bt} = 1.77$ kg.

e) *Wall contour blastholes*

The length of contour that is left for a height of 4.0 m is:

$4.0 - B_z - B_{ct} = 4.0 - 1.14 - 0.62 = 2.24$ m, with $f = 1.2$ and $S/B = 1.25$ one has:
$B_{ch} = 1.33 - L \times \text{sen}\, 3° - 0.05 = 1.12$ m
$NB = |\ 2.24\,(1.33 \times 1.25) + 2\ | = 3$
$S_{ch} = 2.24/2 = 1.12$ m
$l_f = 1.40$ m
$l_c = 1.35$ m
$Q_b = 3.2$ kg.

f) *Stoping*

As the side of the fourth section is $A'_h = 1.42$ m and the practical burden of the wall contour blastholes is $B_{ch} = 1.12$ m, the available space for a tunnel width $AT = 4.5$ m is:

$$4.5 - 1.42 - 1.12 \times 2 = 0.84 \text{ m}$$
$$B = 1.21 - 0.05 = 1.16 \text{ m for } f = 1.45$$

However, $B = 0.84$ m will be used owing to the horizontal dimensions of the tunnel.

For the upper blastholes:

$$B = 1.33 - 0.05 = 1.28 \text{ m}$$

but, if $A'_h = 1.42$, $B_z = 1.14$ and $B_{ct} = 0.62$ is subtracted from the tunnel height, the following is obtained:

$$4.5 - 1.42 - 1.14 - 0.62 = 1.32 \text{ m}$$

As the difference is only 5 cm, B is made equal to 1.32 m.

The charge for stoping holes is the same as for wall holes, thus:

$$Q_b = 3.20 \text{ kg.}$$

g) *Summary*

- Cut: 16 blastholes
  $(4 \times 1.59) + (4 \times 2.62) + (8 \times 3.67) = 46.21$ kg
- Lifters: 5 blastholes (5×3.20) = 16 kg
- Roof contour: 8 blastholes (8×1.77) = 14.16 kg.
- Wall contour: 6 blastholes (6×3.20) = 19.20 kg.
- Stoping: 5 blastholes (5×3.20) = 16.00 kg.

Total charge of the blast = 111.6 kg
Tunnel surface = 19.5 m^2
Advance = 3 m
Volume of broken rock = 58.5 m^3

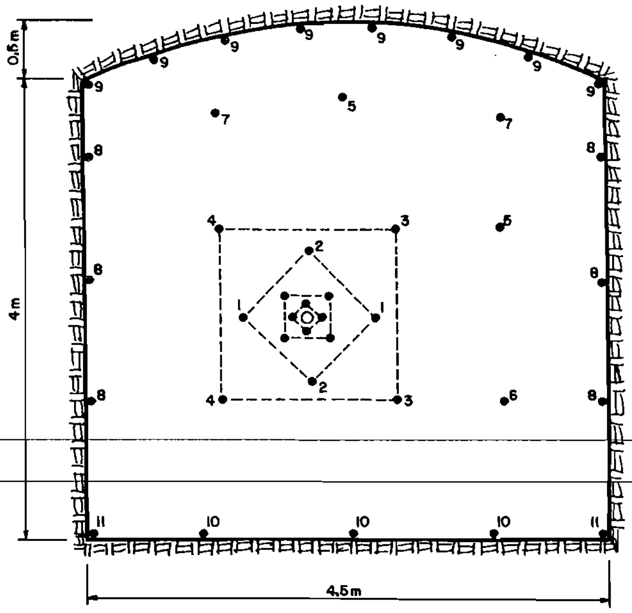


Fig. 22.16. Geometric pattern of the calculated blast.

Specific charge = 1.9 kg/m^3
Total number of blastholes = 40
Total length drilled = 128 m
Specific drilling = 2.2 m/m^3

Simplified calculation

In order to calculate tunnel blasts with parallel hole cuts in four sections more quickly, the equations shown in Tables 22.2 and 22.3 can be applied:

a) *The cut*

b) *Stoping*

In order to calculate the rest of the blast, start from the burden size B and the charge concentration in the bottom q_f for the explosive and diameter used. The formulas used are:

$$q_f = 7.85 \cdot 10^{-4} \cdot d^2 \cdot \rho$$
$$B = 0.88 \cdot q_f^{0.35}$$

where: d = Diameter of explosive cartridge (mm), ρ = Density of explosive (g/cm^3).

Verification of the blast patterns

Once the calculation of the patterns and charges has been done and before doing the blasts, it is interesting to check and contrast the data obtained with the standard or typical results of similar operations. These verifications can be carried out with simple graphics such as those of the Figs 22.17, 22.18 and 22.19, where the powder factor is shown as a function of the tunnel section and drilling diameter, the number of blastholes per round and the specific drilling taken from the two indicated parameters.

The previous graphics refer to blasts with parallel blastholes and only can be taken as a guide, as many parameters influence the results of the excavation: types of rock and explosives, blasthole size, types of cut, need for contour blasts, vibration limitations, etc. which can cause slight variations in the design parameters.

The final verification of the calculations will be made after the blast. The introduction of the necessary modifications after an analysis of the results in the first trials should be gradual and systematic, even to the point of not drilling the holes to their full length in the first rounds and increasing the advance little by little in each cycle.

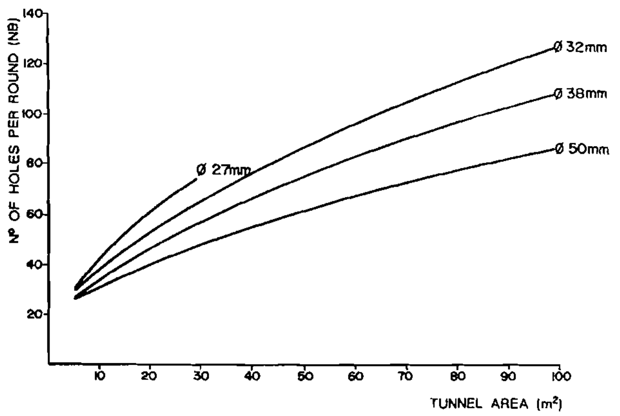


Fig. 22.18. Number of blastholes per round in function of the area.

22.4.2 *Burn cuts*

In these cuts all the blastholes are drilled parallel and with the same diameter. Some are charged with a large quan-

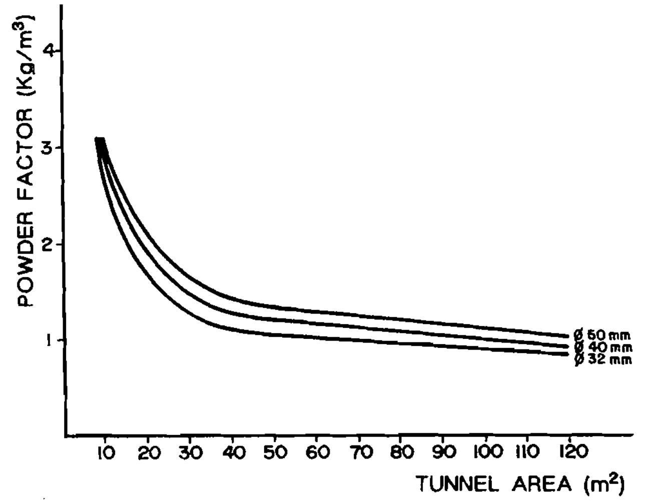


Fig. 22.17. Powder factor in function with the tunnel area (m^2) and the blasthole diameters.

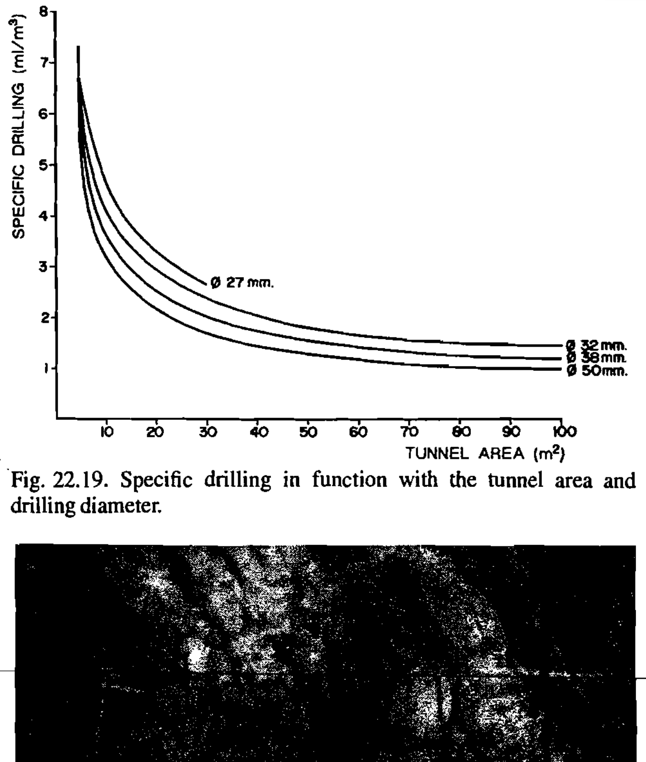


Fig. 22.19. Specific drilling in function with the tunnel area and drilling diameter.

Table 22.2.

Section of the cut	Burden value	Side of the section
First	$B_1 = 1.5\,D_2$	$B_1\sqrt{2}$
Second	$B_2 = B_1\sqrt{2}$	$1.5\,B_2\sqrt{2}$
Third	$B_3 = 1.5\,B_2\sqrt{2}$	$1.5\,B_3\sqrt{2}$
Fourth	$B_4 = 1.5\,B_3\sqrt{2}$	$1.5\,B_4\sqrt{2}$

Photo 22.2. Manual drilling in a drift face.

Table 22.3.

Part of round	Burden (m)	Spacing (m)	Length bottom charge (m)	Charge concentration Bottom (kg/m)	Charge concentration Column (kg/m)	Stemming (m)
Floor	B	1.1 B	L/3	q_f	q_f	0.2 B
Wall	0.9 B	1.1 B	L/6	q_f	$0.4\,q_f$	0.5 B
Roof	0.9 B	1.1 B	L/6	q_f	$0.36\,q_f$	0.5 B
Stoping						
Upwards	B	1.1 B	L/3	q_f	$0.5\,q_f$	0.5 B
Horizontal	B	1.1 B	L/3	q_f	$0.5\,q_f$	0.5 B
Downwards	B	1.2 B	L/3	q_f	$0.5\,q_f$	0.5 B

tity of explosive while others are left empty. As the charge concentration is high, the fragmented rock is sinterized in the deep end of the cut, without presenting optimum conditions for the outcome of the round as happens in spiral cuts. The advances are reduced and do not surpass 2.5 m per round, Fig. 22.20.

One of the burn-cuts that is used in drift advances of coal mines is called *Sarrois Cut*, which is composed of eight charged blastholes and one empty one. With the drilling diameter of 38 mm, the distance between the axes of the blastholes goes from 10 cm in hard rock up to 20 cm in soft rock. This cut is used in depths of up to 2.5 m, with a high powder factor. The charges are designed as shown in Fig. 22.21, avoiding flashover in each of the blastholes with different delay timings and generally using clay plugs for stemming.

The projection of broken rock reaches a length of 5 to 6 meters from the new face and the advances oscillate between 80 and 95%.

Finally, another cut that is also used in coal mines, above all in the north of Spain, is the one called *Swedish*, where the blasthole placement, according to the type of rock, is shown in Fig. 22.22.

For a diameter of 38 mm, the distance between the vertical rows is 20 cm, the vertical separation between blastholes of the two lateral rows is 30 cm and the vertical distance between charged and empty blastholes is from 10 to 15 cm, according to the compressive strength of the rock.

The broken rock projection is better than with the Sarrois cut, although, on the contrary, the powder factor is lower. The advances oscillate between 90 and 100% of the depth and the drilling must be precise.

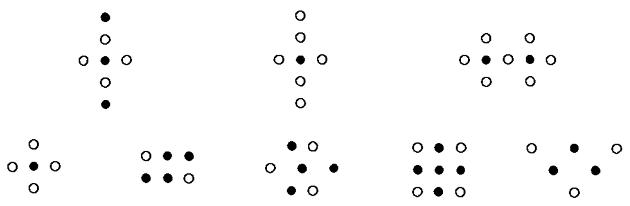


Fig. 22.20. Examples of burn-cuts.

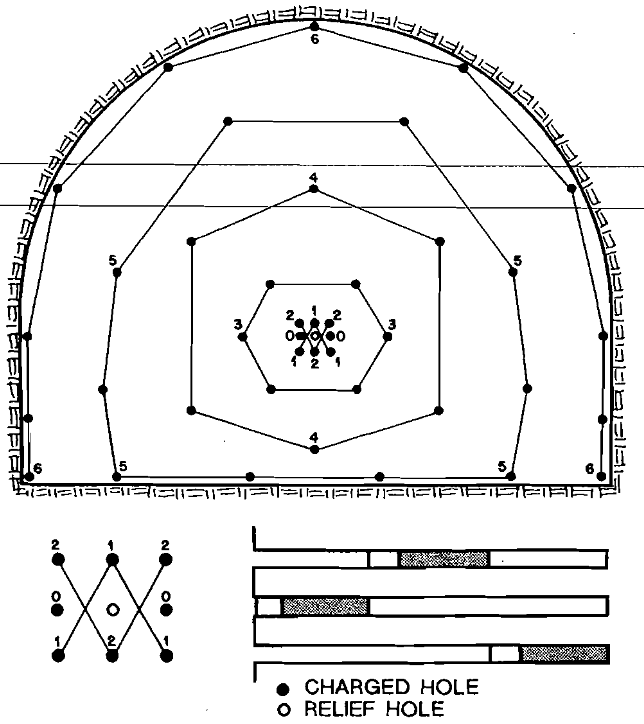
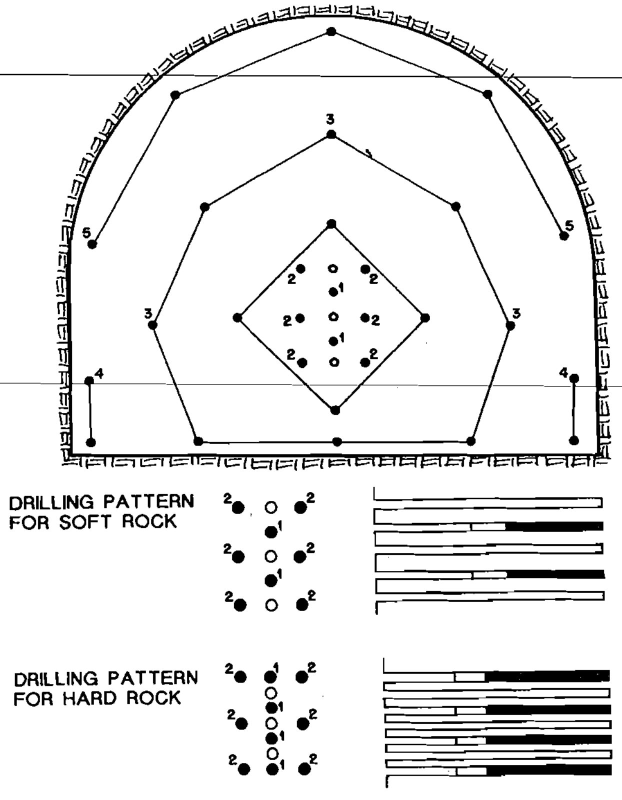


Fig. 22.21. Drift driving with the Sarrois cut.


DRILLING PATTERN FOR SOFT ROCK
DRILLING PATTERN FOR HARD ROCK
Fig. 22.22. Swedish Cut

22.4.3 *Crater cuts*

This type of cut was originally developed by Hino in Japan, taking advantage of the cratering effect that the explosive charges concentrated in the bottom of the blastholes produced upon the nearest free surface.

This method is applied more in shaft excavations than for tunnels, although some specialists such as Hagan have recently suggested their use by placing the concentrated charges in one or various central blastholes of large diameter and distributing the stoping blastholes around the rest of the section with different charge lengths.

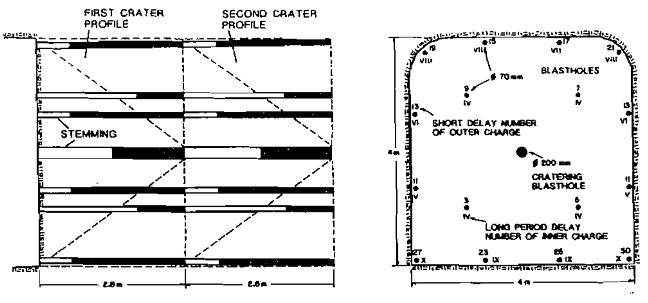


Fig. 22.23. Double crater cut using central blastholes of 200 mm (Hagan, 1981).

As the advance per round is not very large, it has been suggested that the cratering be carried out with double depth blastholes, decking and stemming the charges, Fig. 22.23.

22.4.4 *Angled cuts*

This group of cuts is used less each day because of the difficulty in drilling the holes. The advantage is a lower consumption of explosives because there is better utilization of the free face surface and the possiblity of orientation towards the visible discontinuities in the section.

The following explains the most common angled cuts.

V Cut

In these cuts in wedges or *V*, the advances are around 45 to 50% of the tunnel width. In wide tunnels, the advances are affected by the deviations of the blastholes, which is usually near 5%. Therefore, in a 5 m long blasthole, the deep end could have up to a 25 cm deviation, which might cause sympathetic detonation of the adjacent charge.

The bottom angle should not be less than 60°, because the charges would be too confined and more explosive would be necessary to achieve adequate fragmentation.

The mean parameters in the cut design, in function with the drilling diameter *D*, are the following:

- Total height of the cut $H_c = 46\,D$.
- Burden $B = 34\,D$.
- Bottom charge concentration $q_f = 990\,D^2$ (D in m).
- Length of bottom charge $l_f = 0.3\,L$.
- Column charge concentration $q_c = 0.5\,q_f$.
- Length of stemming $T = 12\,D$.
- Number of wedges in the vertical sense 3.

The *cut-spreader* blastholes, which are also drilled inclined with respect to the tunnel axis, Fig. 22.24, are placed according to the following equations:

- Burden $B = 24\,D$.
- Bottom charge concentration $q_l = 990\,D^2$.
- Length of bottom charge $l_f = 0.3\,L$.
- Column charge concentration $q_f = 0.4\,q_f$.
- Stemming length $T = 12\,D$.

The burden value should fulfill the following condition $B \leq 0.5\,L - 0.2$ m, which suggests that in shallow blasts the burden should be reduced.

The cut holes, or even the nearest *cut-spreader* holes should be fired with milisecond delay detonators and the rest with delay detonators.

The drilling patterns for the stoping, lifter and contour areas are calculated in the same manner as indicated for parallel blasthole cuts.

Fan cut

This type of cut was widely used several years ago, but it is not favored nowadays because of its complicated drilling.

The blasthole patterns and charges for the cut are calculated as in the 'V' cut, Fig. 22.25.

The *cut-spreader* blastholes are measured by the following equations:

- Burden (it should fulfill $B < L - 0.4$) $B = 23\,D$.
- Height of cut $H_c = 42\,D$.
- Concentration of the bottom charge (D and m) $Q_f = 990\,D^2$.
- Length of bottom charge $l_f = 0.3\,L$.
- Concentration of bottom charge $q_c = 0.4\,q_f$.

The initiation sequences of the cut and *cut-spreader* should be carried out with milisecond delay detonators.

The fans can be horizontal, as the former, or drilled upwards or downwards, Fig. 22.26.

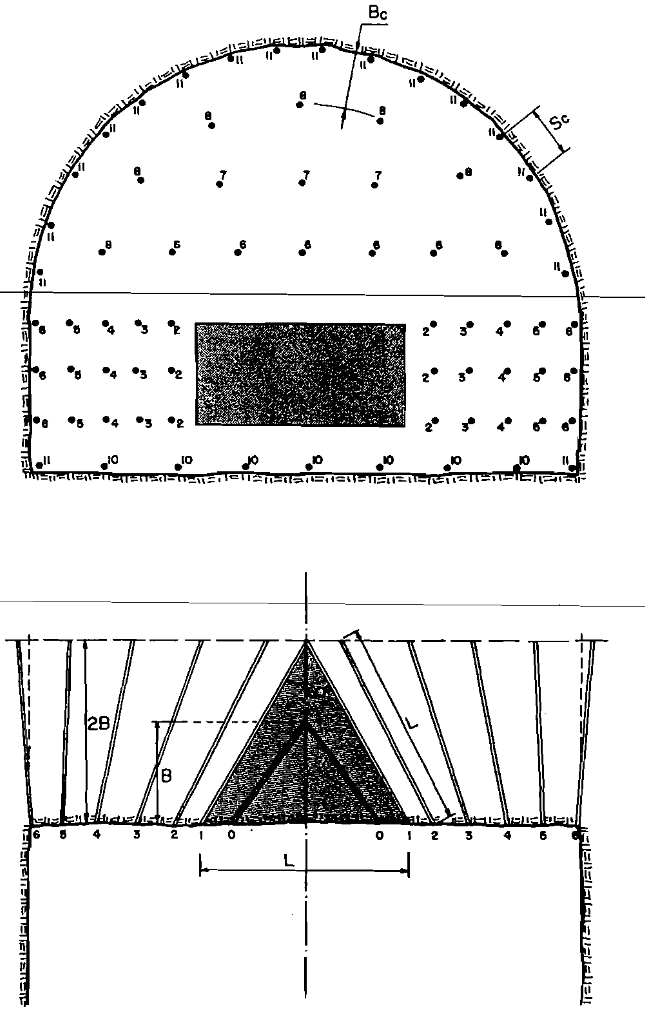


Fig. 22.24. 'V'-cut blast.

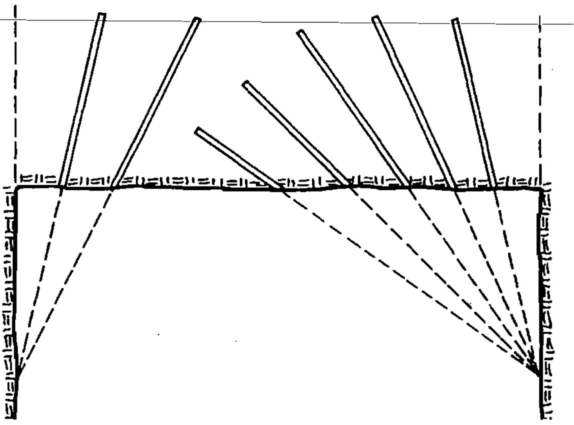


Fig. 22.25. Horizontal fan-cut.

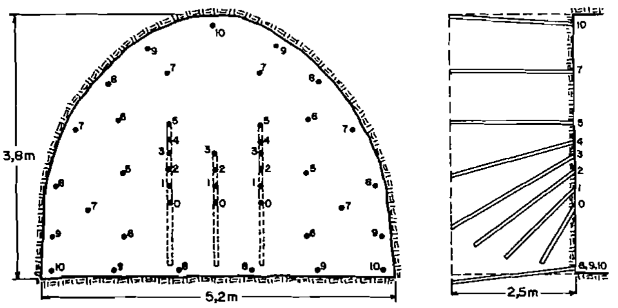


Fig. 22.26. Fan-cut floor blast.

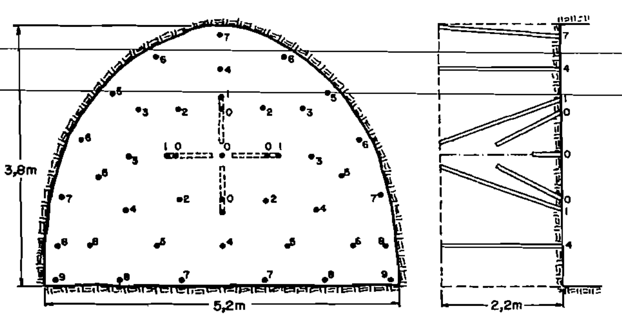


Fig. 22.27. Instantaneous pyramidal cut blast.

Instantaneous cuts

One of the variations of the *V* cut is to drill a cluster of adjacent blastholes and fire all the charges simultaneously. Advances of around 80% of the tunnel width can be obtained.

One of the inconveniences of these cuts is the great projection of the broken rock which disperses it at a considerable distance from the tunnel face.

Between the variations that exist, the pyramidal cut with one or two sections is worth mentioning, Fig. 22.27.

22.4.5 *Drifts with coal beds*

The advance blasts for drifts with coal beds on the face can be quite varied, according to the excavation sections, bed strengths, inclination, face disposition, etc., therefore only a few general considerations will be made.

The drilling patterns should be parallel to the stratification, with all the charged blastholes breaking towards the free space created in the coal bed. These cuts or cavities, can be drilled manually if the coal is soft or, as is usually done, by firing some blastholes upon the coal itself with a low delay number, Fig. 22.28. This last procedure has the inconvenience of mixing the coal with the waste, preventing its use, but it is the method which gives the most advance.

In the beds with sudden detachments of firedamp, it is recommended to leave an uncharged hole to degasify the coal. According to labor classification, because of the existance of gases, dust and other explosive or inflamable substances, the maximum explosive charges per blasthole will be determined along with the type of explosive, the length of the round and the maximum delay timings between holes.

Spanish legislation, through the ITC 10.4-10, establishes a classification of the labors in mines of second and third category, and for those labors in which the presence of gases, dust or other explosive or inflammable substances is possible, as indicated in Table 22.4.

In the mentioned Table, the type of explosive, the maximum amount per hole, the type of detonator and the maximum deviation of the blast are indicated.

22.4.6 *Drifts in salt mines*

In the sedimentary beds of soft minerals such a salts, potash, etc., the drifts to prepare the cuts can be excavated

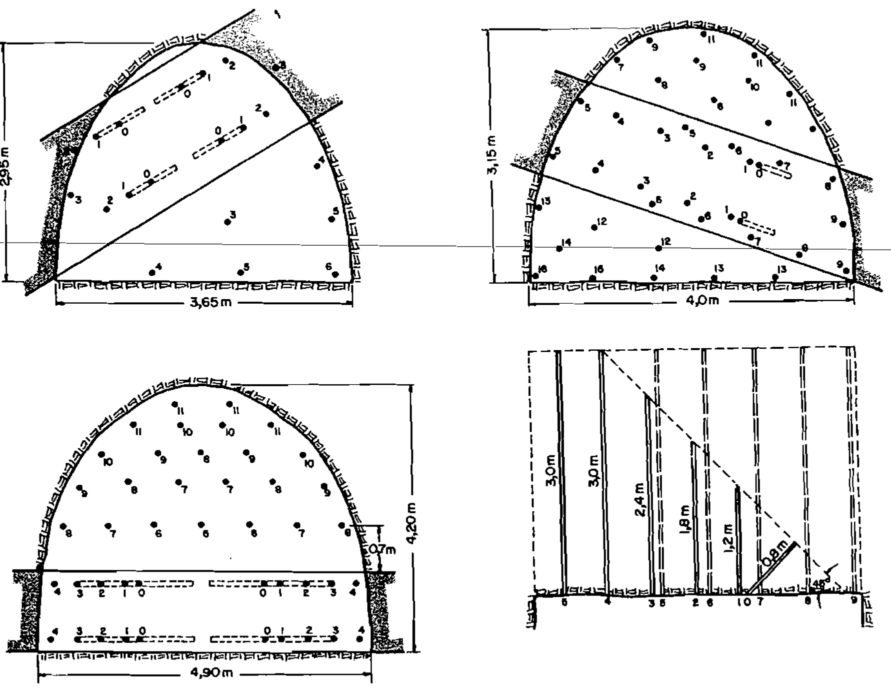


Fig. 22.28. Blasts in drifts with coal beds.

Table 22.4.

LABOR	TYPE	CONDITIONS	EXPLOSIVE	MAXIMUM AMOUNT PER HOLE IN GRAMS	DETONATOR	MAXIMUM DURATION OF BLAST	OBSERVATIONS
LABOR IN ROCK - ROCK DRIFT - CROSSCUT	1ª	- That the face does not go through coal - That the holes do not go through coal - In horizontal or upward work the fire damp concentrations at the face and last 100 m be under 0,5% - If the work is carried out with air from other labors that contains fire damp it cannot 60 over 1% - In upward labors, the maximum conceruration can never de over 0,5% - In the last 30 m. there cannot be any accumulation of coal or dust, nor work faces or coal haulage drifts. - That in the last 30 m the layers of incovered coal be under 10% of the total surface of the labor in tha area and that the last layer be at least at 3 m.	UNLIMITED	NO LIMIT	DELAY OR MILLISECOND DELAY		
ROCK DRIFT	2ª	- That the face not go through coal - That the not of holes that go througt coal be under 1/5 of the total. - Mixed labors of coal and rock in which the total area of coal does not exceed 10% of total surface or those in which the number of holes which go through coal do not exceed 1/5 of total	SAFETY, 9 SAFETY, 9 bis SAFETY, 12	2.000	DELAY OR MILLISECOND DELAY	5 s	BLASTHOLES THAT HAVE CUT THROUGH COAL CANNOT BE CHARGED
ORE DRIFT	3ª	- Mixed labors of coal and rock in which the total area of un covered coal be over 10% of total surface.	SAFETY, 9 SAFETY, 9 bis	1.000	MILLISECOND DELAY MAXIMUM 7 nº of 20 ms 5 nº of 30 ms	125 ms	
		- That the number of holes that went througt coal be over 1/5 of total	SAFETY, 12				
WORK FACE	4ª	- Labors over bed/which are ventilated by the main air lines	SAFETY, 9 SAFETY, 9 bis SAFETY, 12	500	MILLISECOND DELAY 7 nº of 20 ms 5 nº of 30 ms	125 ms	
			SAFETY, 20 SR SAFETY 18 SR	2.000	MAXIMUM DELAY	5 s	WITH AIR VELOCITY V ≥ 0,5 m/S
			SAFETY 30 SR	2.500	MILLISECOND DELAY	500 ms	WITH AIR VELOCITY V < 0,5 m/S
	5ª	- Labors over bed/not ventilated by main air flow	SAFETY 20 SR SAFETY 18 SR SAFETY 30 SR	2.000 2.500	MILLISECOND DELAY MILLISECOND DELAY	↙ 500 ms ↖ 500 ms	
	6ª	- Breakage of coal masses/unventilated.	SAFETY 20 SR SAFETY 18 SR SAFETY 30 SR	1.500 2.000	MILLISECOND DELAY MAXIMUN 7 nº of 20 ms 5 nº of 30 ms	125 ms 125 ms	

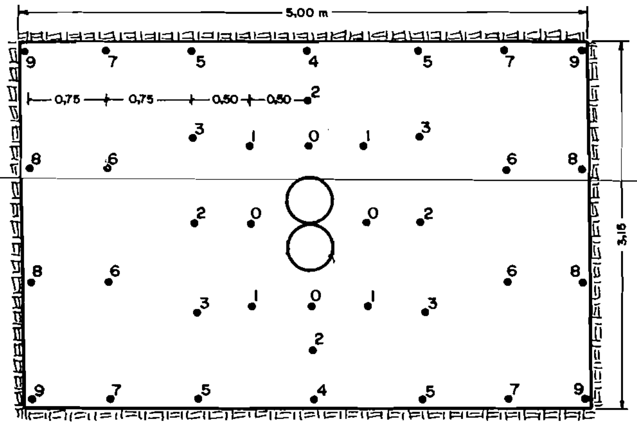


Fig. 22.29. Drilling pattern with spiral cut composed of two large diameter blastholes.

by drilling and blasting, as well as with continous miners.

The drilling is usually carried out with jumbos that are capable of opening cut holes with a diameter of up to 420 mm and a depth of 7 m, Fig. 22.29. The rest of the blastholes with diameters of 37 and 42 mm, are usually drilled parallel to the axis of the tunnel and with the same depth as those of the cut. The explosive charging should be mechanized because, otherwise, the great length of the holes makes it very difficult.

If the modus operandi is chambers and pillars, the opening of the chambers can be done with a central drift and lateral stoping holes for widening. All drilling is done horizontally, as indicated in Fig. 22.30.

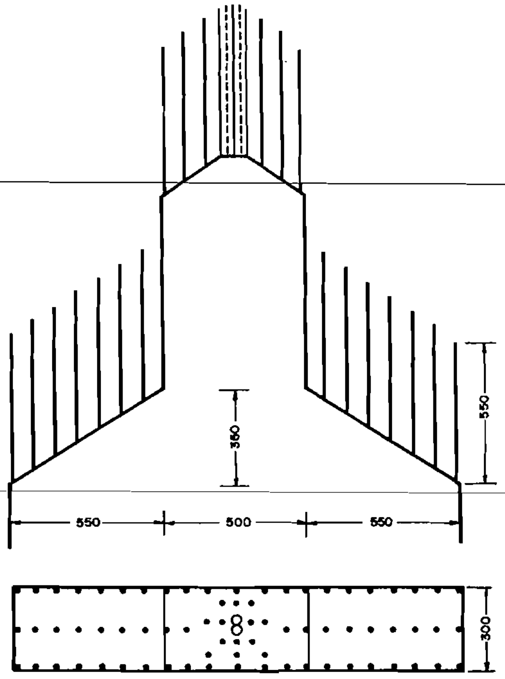


Fig. 22.30. Advance sequence for an excavation of chambers and pillars.

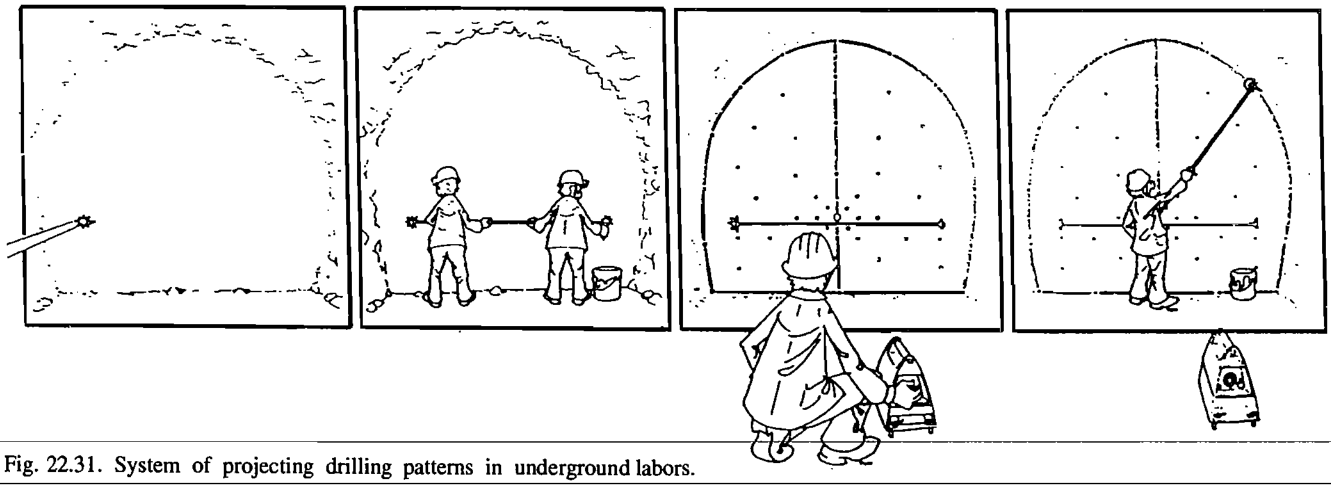


Fig. 22.31. System of projecting drilling patterns in underground labors.

22.5 EQUIPMENT FOR MARKING OUT DRILLING PATTERNS

Amongst the auxiliary equipment for marking out the collars of the blastholes in underground labors, the drilling pattern projectors are available. These units are battery run and can be placed on a tripod on the ground or upon a vehicle. Once the direction of the tunnel or shaft has been marked, two reference points are indicated on the face and, following this, the hole pattern of the round is projected on the rock. The obtained image is focused and the collaring points of the blastholes are marked with paint, Fig. 22.31.

REFERENCES

Berta, G.: *L'Explosivo Strumento Di Lavoro.* Italexplosiv, 1985.

Cil.: *Manuel del Explosifs.* Montreal, 1971.

Du Pont: *Blaster Handbook*, 16th edition, 1980.

EXSA: *Manual Práctico de Voladura*, Perú, 1986

Gregory: *Explosives for North American Engineers.* Trans Tech Publications, 1983.

Godoy, S.G. & M.D. Viera: *Computerized model for design optimization of blasting patterns in tunnels.* Tunnelling '82, 1982.

Gustafsson, R.: *Swedish Blasting Technique.* SPI, 1973.

Hagan, T.N.: *Larger diameter blastholes – A proposed means of increasing advance rates.* Fourth Australian Tunnelling Conference Melbourne, 1981.

Herrmann, K: *Precis de Forage des Roches.* Dunod, 1971.

Hempfill, G.B.: *Blasting Operations*, MacGraw-Hill, 1981.

Holmberg, R.: *Charge calculations for tunnelling.* Underground Mining Methods Handbook, AIME, 1982.

Larsson, B. & D.A. Clark: *Cost savings and improved stability through optimized rock blasting.* VME-Nitro Consult, 1982.

Langefors, U. & B. Kihlstrom: *Voladura de Rocas.* Edit. URMO, 1973.

Lopez Jimeno, C.: *Cálculo y Diseño de Voladuras en Túneles con Microordenador – Programa DISVOLTUN.* I Seminario de Ingeniería de Arranque con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, Marzo, 1986.

Olofsson, S.O.: *Applied explosives technology for construction and mining.* APPLEX, 1990.

Prinz, J.: *Perspectives D'Avenir de Creusement a l'Explosif.* Industrie Minèrale – Les Techniques. Août-Septembre, 1983.

Rockset Int. Salej A.B.: *Información Técnica.*

Rustan, A. et al.: *Controlled blasting in hard intense jointed rock in tunnels.* CIM Bulletin, December, 1985.

Tamrock: *Handbook of Undergroung Drilling.* 1983.

Wild, H:W.: *Sprengtechnik.* Verlag Glükauf. 1984.

CHAPTER 23

Shaft sinking and raise driving

23.1 INTRODUCTION

In any project that profits from the underground such as ore recovery or the opening of large chambers, an essential part of the development work is the driving of shafts and raises. These can be either vertical or inclined, and are characterized by their lineal design and drilling difficulties.

During the last decades numerous methods have been developed, mechanizing the work by means of special techniques and drilling aparatus, increasing advances and production and improving safety conditions.

23.2 SHAFT SINKING

The methods for shaft sinking can be divided in three groups:

- The benching method (half-bottom method).
- The spiral method.
- The full-bottom method.

23.2.1 *The benching method*

This method is particularly suitable for square or rectangular sided shafts. It consists in drilling one half of the shaft cross section with a fan, leaving the other half (lower) as a sump for water or spoil, if necessary, or as a free cavity.

Blasting is similar to that in small benches with a free face, displacing the broken rock towards the space left by the former round, Fig. 23.1.

Drilling is usually done by hand or with pneumatic hammers.

23.2.2 *The spiral method*

This was first used in Sweden and consists in excavating downwards in a spiral, with a height that depends upon the diameter of the shaft and the type of ground to be fragmented. Within each cut, a section of the spiral is blasted with an angle large enough to make the time necessary to carry out a complete cut coincide with a whole unit of the available work time, Fig. 23.2.

The blastholes in each radius are drilled parallel and of the same length, as there always has to be a free face which descends with each position.

Apart from the advantages of the yield and costs of this method, other interesting aspects are that the length of the cut can be synchronized in function with the work plan, the drilling patterns and blasting systems are simple and the drilling personnel does not have to be very experienced. Lastly, as the broken rock always remains in the deepest part, the loading equipment works with a high productivity.

23.2.3 *The full-bottom method*

The full-bottom method is used frequently in shaft sinking, as it suits either rectangular or round sections shafts.

There are various techniques for blasthole placement because, as happens in drifting and tunnelling, it is necessary to create a free surface with a few blastholes unless there is a large diameter pilot hole available or expansion raises. The face is opened with plough cuts, V-cuts or cone cuts and with relief blastholes, Fig. 23.3.

V-cuts are used in rectangular section shafts. The planes of the dihedrals formed by the blastholes that are inclined between 50 and 75° should be parallel to the discontinuities, in order to use them to advantage during breakage.

Cone cuts are used most in round-section shafts due to, on one hand, the ease with which blasthole drilling can be mechanized and, on the other, the lower powder factor when compared with the plough cut. The holes are placed so as to form several inverted cone areas in the central part, as shown in Fig. 23.4.

The pull of the rounds, as well as the number of blastholes, depends upon many parameters such as: type of ground, diameter of the explosive charges, the blasting pattern, type of cut, work plan, and, above all, the section of the excavation.

Set rules cannot be given for the two design parameters mentioned because they need to be adjusted in each case. For example, the number of blastholes needed for the driving of several shafts in South Africa, with 32 mm diameter charges, can be estimated by using following equation (Espley-Jones and Wilson, 1979):

$$NB = 2\,D_p^2 + 20$$

where: NB = Number of blastholes, without including those of the perimeter, if contour blasts are carried out,
D_p = Shaft diameter (m).

As to the drilling depth in each round, Wild (1984) recommends starting from the values indicated in Fig. 23.5.

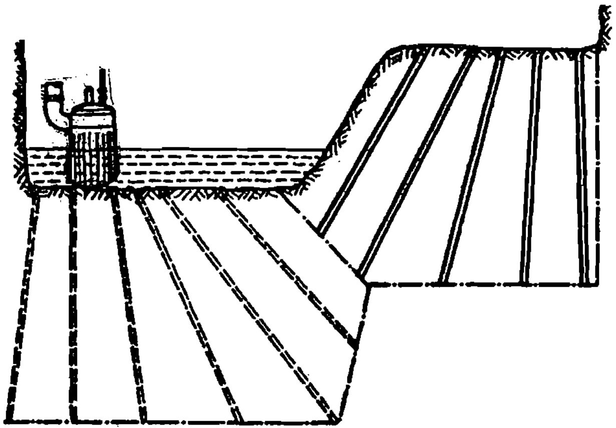


Fig. 23.1. The benching method.

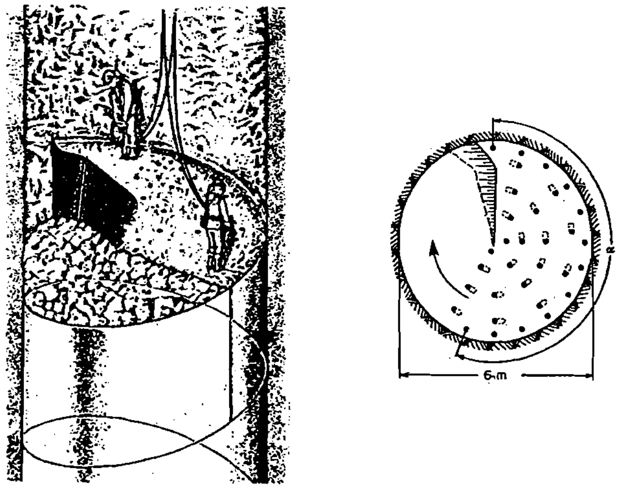


Fig. 23.2. The spiral method.

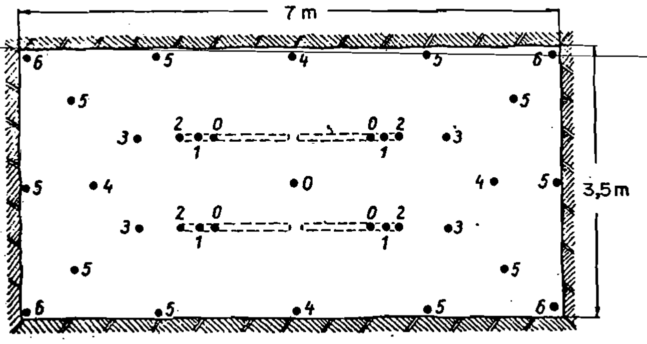


Fig. 23.3. Drilling pattern in a rectangular-section shaft.

The same author provides a graphic on the powder factor in function with the dimensions of the shafts, Fig. 23.6

The plough cut works as in tunnels and drifts, Fig. 23.7. The results obtained up to now are interesting, giving the additional advantage of easier drilling.

A variation of the former consists in blasts with a large diameter central blasthole or raise. In these cases, there is a more efective free face which favors breakage and rockthrow, as well as loading, Fig. 23.8.

If cartridged explosives are used, the relationship between the blasthole diameter and the caliber of small cartridges should be around 1.2 to 1.25, or have a space of about 10 mm for the larger ones. The use of bulk slurries is ideal for reducing the number of blastholes or for taking maximum advantage of the drilling.

The detonators are usually connected in parallel, placing the circuits in ring shape, Fig. 23.9.

23.3 RAISE DRIVING

Raises are excavations with reduced dimensions and an inclination that is over 45°. Raise driving is a typical operation in mining and the lengths can vary, up to more than 100 m. Raises are used to unite drifts on different levels closing the ventilation circuits, for the passage of ore and spoils, for the initial openings of sublevel stoping, etc. Raise driving is also frequent in civil engineering, especially in hydraulic plants and underground warehouses.

The driving of raises has been, up until recently, one of the most difficult operations in rock breakage by drilling and blasting, until the long-hole method came into use.

Raise driving is classified in two large groups according to the drilling method used, either upward or downward:

– *Upward Drilling.* Done by hand with a Jora lift or with the Alimak platform.

– *Downward Drilling.* Long-holes with pilot hole cut, with crater cut, 'VCR' cut (Vertical Crater Retreat), and the full face method.

23.3.1 *Methods of upward drilling*

These methods were the traditional and only ones in existance until the long-hole method appeared.

Classical manual method

This is applied in small operations where the amount of work to be carried out does not justify investing in special equipment and the raises are not very long.

The method consists in assembling and disassembling an inner wooden platform, simultaneously with the advance of the excavation, which not only acts as support but is also a work platform from which the blastholes are drilled with hand-held jackhammers and pushers. These structures are installed from service ramps as shown in Fig. 23.10.

The blastholes are usually placed in *V* or in fan with pulls per round of 1.5 to 2 m and initiation sequences as shown in Fig. 23.11.

This method is competetive in small mines but the work conditions are difficult and require very experienced personnel.

JORA lift method

This system was discussed earlier in the chapter on special drilling equipment. It consists in a lift that is suspended by a rope that passes through a pilot hole driven previously along the line of the raise.

The blastholes are usually drilled parallel using the

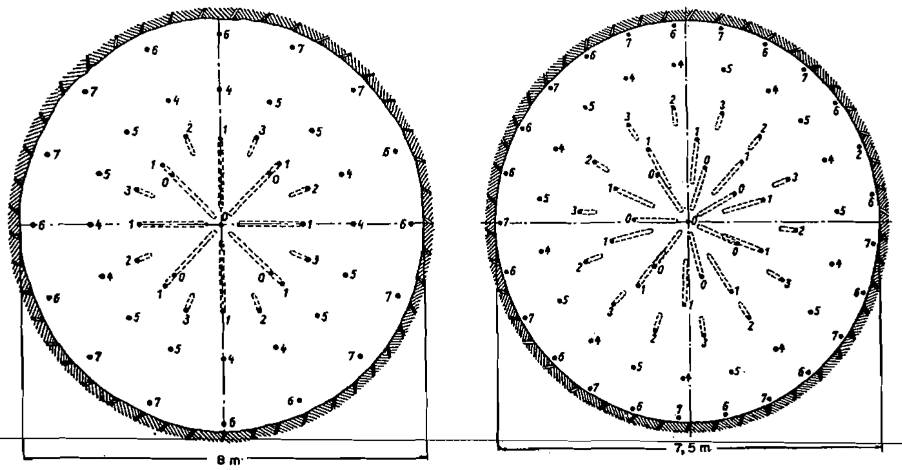


Fig. 23.4. Cone-cut drilling patterns.

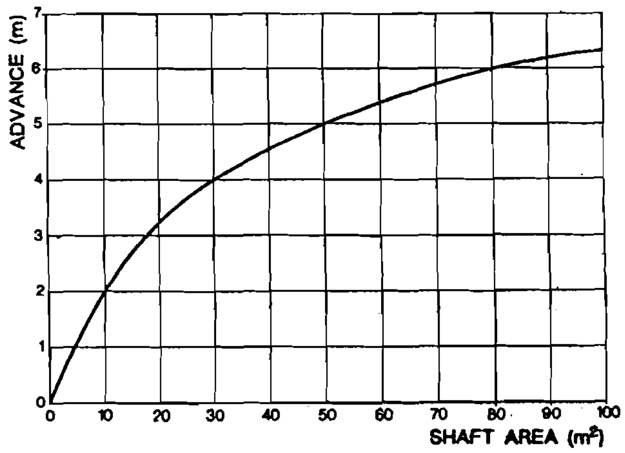


Fig. 23.5. Round depths as function of the shaft section.

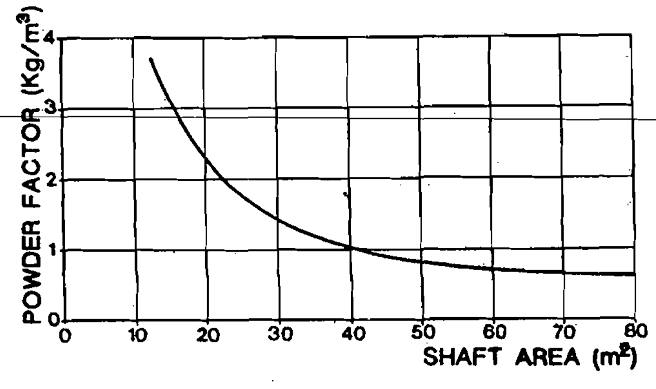


Fig. 23.6. Powder factor as function of the shaft section.

central hole with the largest diameter as cut hole, blasting 3 to 4 m per round, Fig. 23.12.

The central blasthole, apart from serving as cut hole in the blast, also provides ventilation.

Alimak platform method

This consists in a work platform that slides along a rack and pinion rail that is attached to the raise wall by expansion shell bolts.

Parallel blastholes are drilled with heavy hammers and pushers, getting advances per round of up to 3 m. In Fig. 23.13 a typical firing pattern is indicated.

Once the round is fired, the bottom of the raise is ventilated by injecting compressed air and water. Later the platform is lifted and the roof is scaled, reinitiating the work cycle again.

23.3.2 *Methods with downward drilling*

The previous methods have the following inconveniences:

- The complete cycles are very long, drilling, blasting, ventilation and scaling, which means low productivity due to lost time.
- The need for an elevated number of highly qualified personnel.

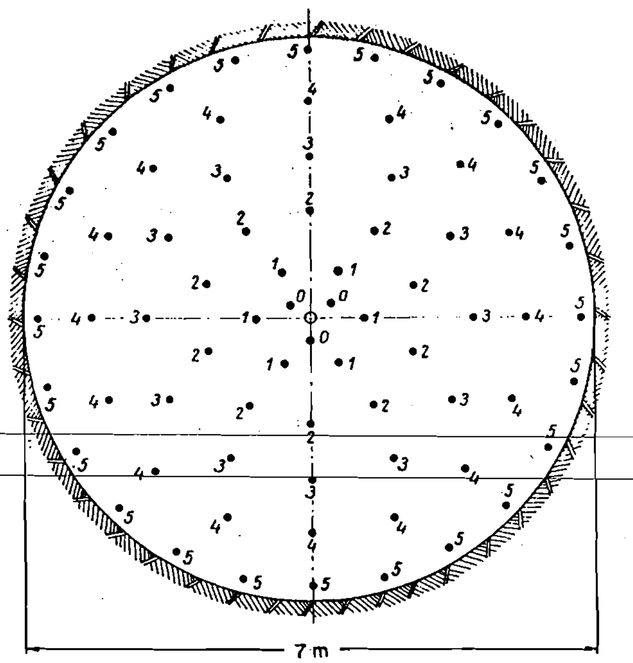


Fig. 23.7. Plough-cut holes.

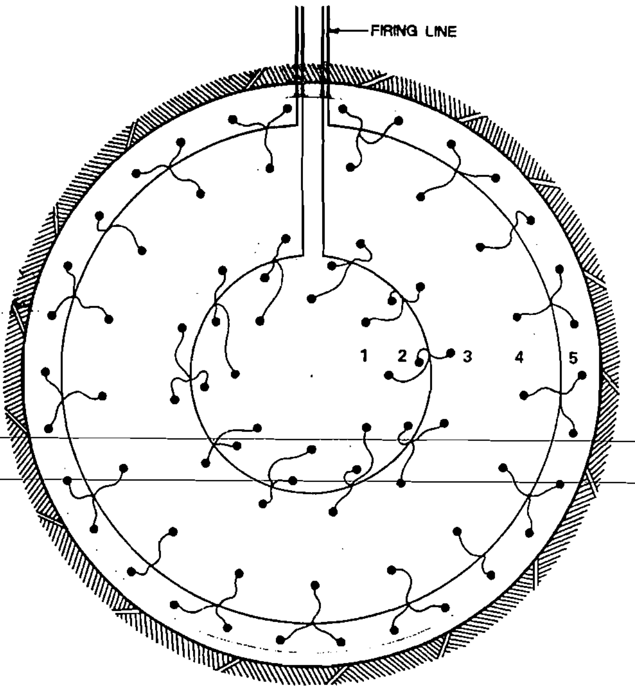


Fig. 23.9. Ring connections in a shaft blast.

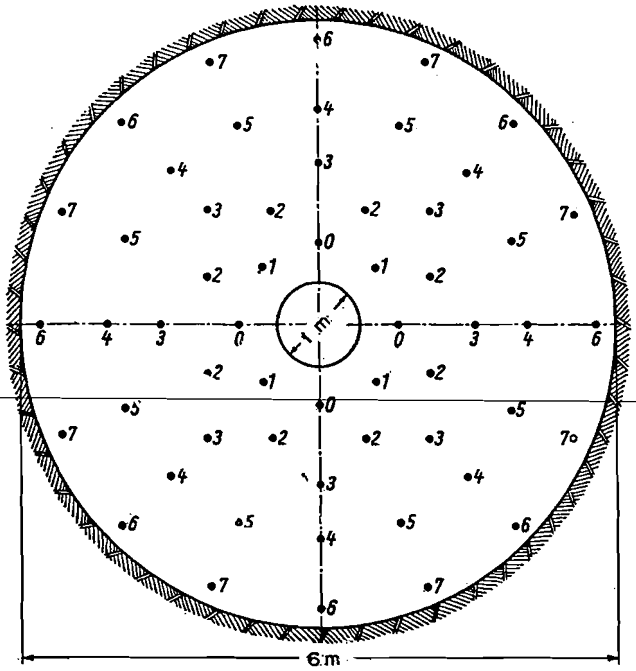


Fig. 23.8. Blast with central raise.

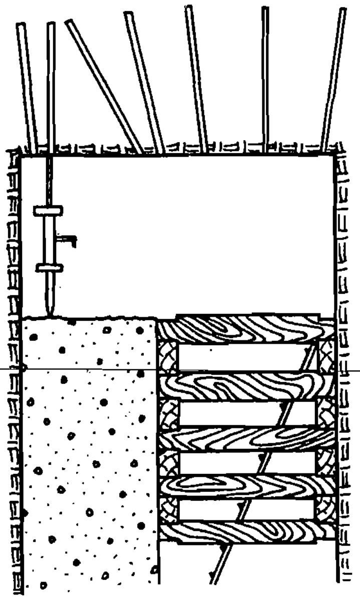


Fig. 23.10. Manual raise building.

- Safety and hygiene conditions are not very good.
- The cost of the operation is usually high.

To solve these problems, in the decade of the seventies various experiments were started by drilling the blastholes to the total length of the raise and later firing the blasts in stages with hanging charges.

Logically, these methods require great drilling accuracy, which motivated the manufacturers to design special equipment and accessories. Nowadays, the deviations can be maintained below 2% with top hammer drills and below 1% with down the hole drills.

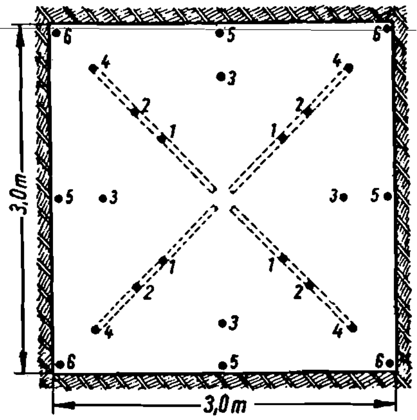


Fig. 23.11. Drilling pattern and initiation sequence in manual raise building.

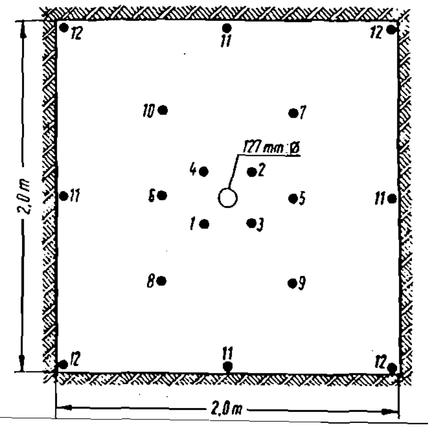


Fig. 23.12. Design of a blast with a large diameter central blasthole.

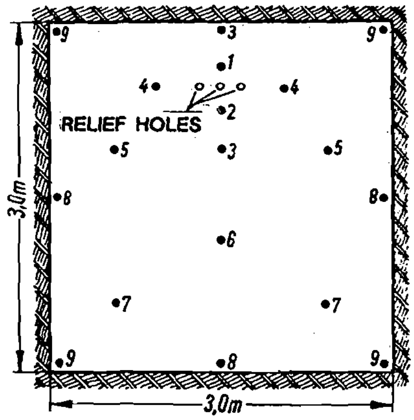


Fig. 23.13. Pattern of a shot with parallel blastholes.

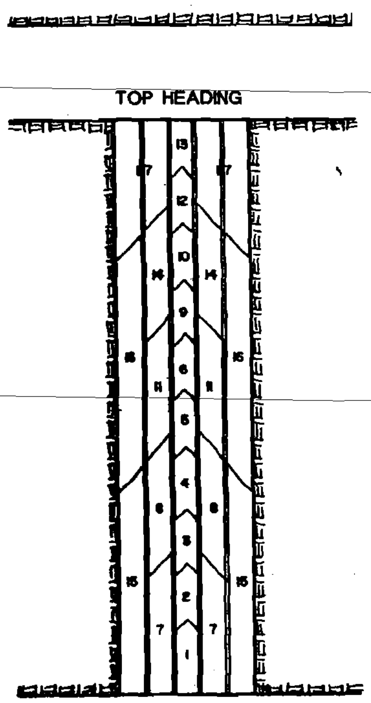


Fig. 23.14. Advance sequence in a raise.

Relief blasthole cut method

This technique, which was developed in tunnelling and drifting, was the first to be applied in raises with longholes.

The holes are drilled with top hammer rigs with diameters between 51 and 75 mm, widening the central pilot holes up to 100 or 200 mm in diameter.

The blastholes are placed, generally, in square-sections which are fired in stages in spans of 2 to 4 m; first the cut zone and then those of the stoping zones, Fig. 23.14, although, if the engineer is experienced, it is possible to do a full-section blast by using milisecond delay detonators in the cut and delay detonators in the stoping holes.

Excessive confinement of the charges should be avoided in order to prevent sinterization of the rock. The lower ends of the blastholes are closed with any of the methods described in the Chapter on Blastings and Production in Underground Mining, and it is recommended to use water as stemming in order to eliminate stoppage.

The patterns for the blastholes of the cut can be calculated with the following equation:

$$S = D_1 + 1.25 \times D_2$$

where: S = Spacing between blastholes, D_1 = Diameter of the charged blastholes, D_2 = Diameter of the central relief hole.

And the blastholes of the stoping sections with the following equation:

$$S = 10 \times D_1 + 500$$

In each section it is suggested that the burden not be larger than the width of the central hole towards which each blasthole breaks.

The lineal charge concentrations in the cut and stoping blastholes are determined with the two following equations:

$$q_1 = 0.03 \times D - 0.85$$

$$q_1 = 0.0125 \times D_1 + 0.26$$

where: q_1 = Lineal charge concentration (kg/m), q_l = Blasthole diameter (mm).

Fig. 23.15 shows the firing pattern for the holes of the cut and the first stoping section in a raise where the central relief hole is of 150 mm.

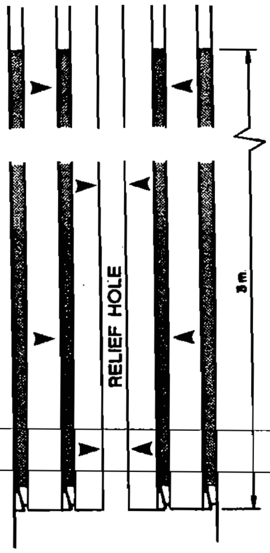
The contour of the raises can be outlined by smooth blasting and placing the blastholes with an average spacing of 13 D.

Crater cut method

This consists in opening a cavity of approximately 1 m^2 with five blastholes of 65 to 102 mm in diameter and placing the explosive charges so that they perform as in crater blasting, Fig. 23.16.

Once the cut has been accomplished in all its length, the stoping is carried out using the patterns and charges indicated for the previous method.

The configuration and situation of the charges is determined by the Livingston theory:



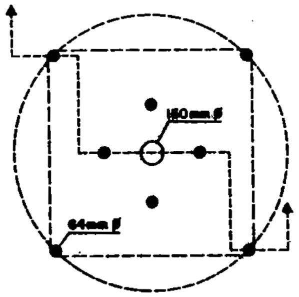


Fig. 23.15. Drilling pattern and initiation sequence in a raise with a large diameter central blasthole.

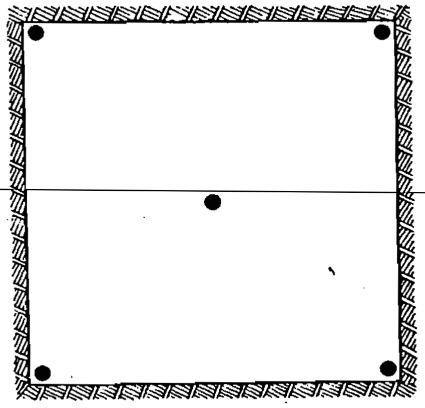


Fig. 23.16. Drilling pattern in the crater cut method.

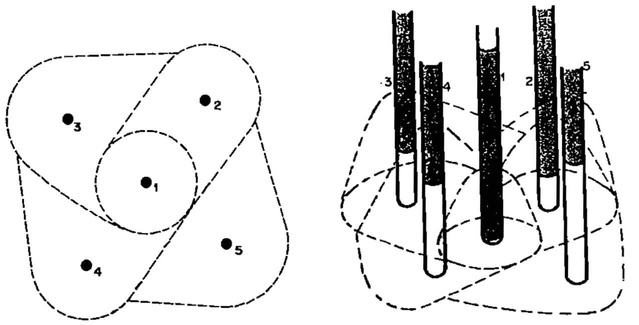


Fig. 23.17. Initiation sequence and distance from the charges to the free face.

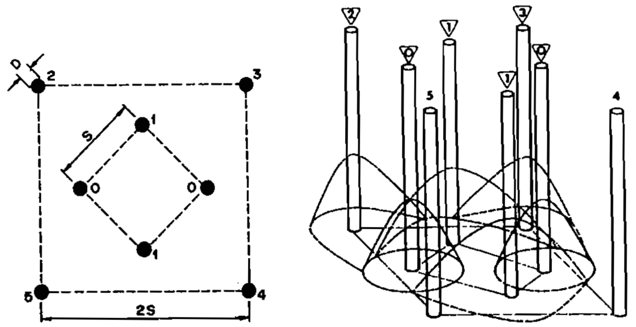


Fig. 23.18. Raise blasting pattern with the 'VCR' method.

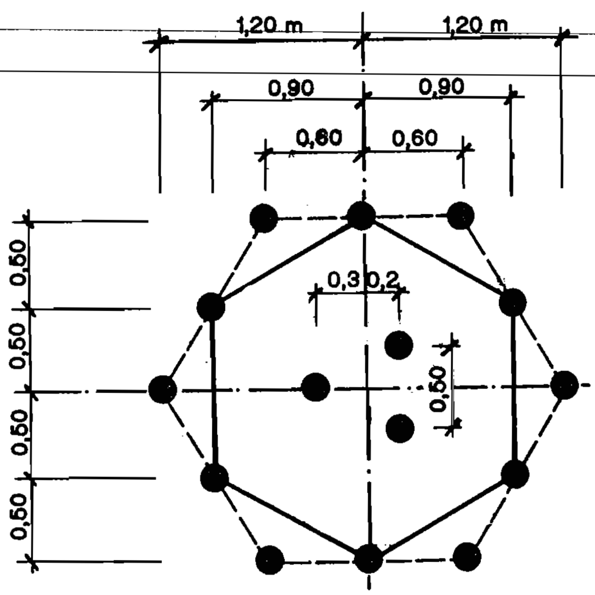


Fig. 23.19. Drilling pattern for driving a raise with a diameter of 2 m, using 165 mm blastholes.

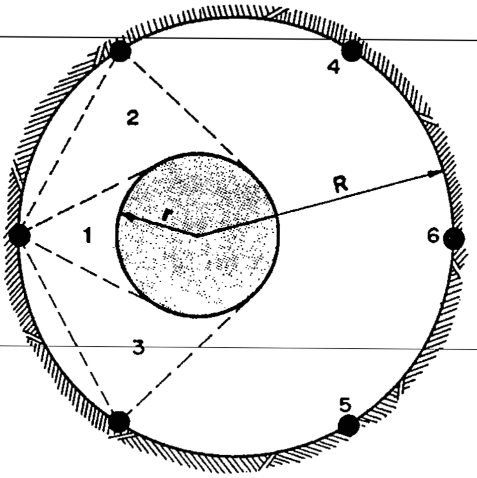


Fig. 23.20. Firing of a blast towards a large diameter pilot hole.

Table 23.1.

	Blasthole Diameter (mm)	
	114	165
Raise section	2.40 × 2.40	3.60 x 3.60
Spacing – S (m)	1.20	1.80
Watergel charge per hole – Q_b (kg)	12	30
Stemming Length – T (m)	1.5	1.8
Advance per round – X (m)	2.10	3.0

– The length of the explosive column should be under 6 D, so that they perform like spherical charges.

– The optimum charge depth is approximately 50% of the critical depth:

$$D_o = 0.5\,D_c$$

– According to the Livingston theory, the critical depth has a value of:

$$D_c = E_t \times Q^{1/3}$$

Where: E_t = The Strain-Energy factor, Q = The explosive charge (kg).

– The quantity of charge Q in the blasthole for an explosive density of ρ_e, has a value of:

$$Q = \frac{3 \times \pi \times D^3}{2} \times \rho_e$$

and, taking the average value of $\rho_e = 1.3$ g/cm^3, it is proven that the optimum depth in function with the blasthole diameter is approximately:

$$D_o = 13.7\,D$$

D_c is the distance between the free face and the center of gravity of the charge in the central blasthole. In the rest of the holes, the depth increases in intervals of some 10 to 20 cm. The blastholes should not be too close together in order to avoid rock sinterization.

The advantages of the crater cut system in comparison with the plough cut are the following:

– Lower drilling costs as there are fewer blastholes and the central hole need not be widened by reaming, and

– Drilling does not have to be as precise.

'VCR' method

At the same time that the Vertical Crater Retreat method became popular, in metal deposit operations a 'VCR' system of driving raises was developed, based on the same principles as the crater cut method, Fig. 23.18.

In this case the blastholes, with diameters similar to those used in production blasting, are placed in square sections with the all charges at the same height. Table 23.1 shows two real examples given by Lang (1981), using high density watergels.

The advantages that this method has over the preceding one are:

– Lower drilling costs and fewer blastholes.

– Easier charging of explosive.

Table 23.2.

Drilling diameter	165 mm
Distance from charge to chamber roof	1 m
Charge height	1 m
Charge height/Diameter relationship	6
Total explosive charge	21 kg
Type of explosive used	Cartridged watergel
Number of cartridges	3 (7 kg)
Advance per round	3 m

Photo 23.1. Vertical excavation of a stoping hole using the free surface of a raise drilled by a Raise-Borer.

– Raise driving all in one phase which means less drilling, and

– The possibility of using down the hole hammer drilling rigs.

Fig. 23.19 shows the drilling pattern used in the Rubiales Mine for raise driving, with 165 mm (6½″) blastholes. The pattern is composed of two hexagonals and an inner triangle. These raises are used for exploitation of the chambers, shooting the production blasts towards the face opened by them.

In Table 23.2, the most interesting data of these blasts has been compiled.

Full-face method

This consists in opening a pilot hole of 1 to 2 m in diameter with a raise borer and using it as an expansion hole.

The method is carried out in large underground civil engineering projects and in shaft sinking or for driving large section raises.

Its principal advantages are:

– Wide drilling patterns which lowers cost.

– Relatively small explosive charges which means less damage to the remaining rock.

– Possiblilty of driving the raise in one shot.

In Fig. 23.19, the pattern and initiation sequence of a blast with this method are shown. The distance of the first blasthole to the pilot hole should be small because the free face is concave and the rock is in good condition after being opened mechanically.

REFERENCES

Atlas Copco: *El Cuele Cráter en la Perforación de Chimeneas.*

Castañon, C. & E. Castells: *Métodos de Perforación y Voladuras Empleados en la Mina de Rubiales.* IV Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, 1990.

Espley-Jones, R.C. & J.C.B. Wilson: *South African blasting practice in shaft sinking.* SEE, 1979.

Gustafsson, R.: *Técnica Sueca de Voladuras.* SPI, 1977.

Lang, L.C.: *Driving underground raises with VCR.* SEE, Sept. 1981.

Lopez Jimeno, E. & C. Lopez Jimeno: *Las Voladuras en Cráter y su Aplicación a la Minería.* VII Congreso Internacional de Minería y Metalúrgica, Barcelona, 1984.
Wild, H.W.: *Sprengtechnik*, Verlag Glükauf GmbH, 1984.

CHAPTER 24

Underground production blasting in mining and civil engineering

24.1 INTRODUCTION

The principal methods for underground mining operations are defined in Fig. 24.1.

The criteria which must be kept in mind when selecting a method are those which refer, on one hand, to the morphology of the deposit and the grade distribution, and on the other hand, the geo-mechanic properties of the rock mass, taking into consideration the mineralization as well as the host rock, and also the technical and economical aspects that each one offers in the conditions of the operation in question.

In this chapter, the principal mining methods used today will be revised, placing special emphasis upon those that have been recently developed using large diameter blastholes (100 to 200 mm), enabling a high degree of mechanization to be reached along with high output and low operational costs. In Table 24.1, the mean productivity per day for each of the methods is shown.

Finally, the procedures for excavation large chambers or caves for use of underground space in non-mining applications, such as hydraulic power plants, liquid fuel tanks, toxic and radioactive residues, etc.

24.2 CRATER RETREAT METHOD

24.2.1 *Crater blasting*

The concept and development of crater blasting attributed to C.W. Livingston (1956), opened a new school of thought for better understanding of the phenomenon of blastings and the characterization of the explosives.

A few years later, Bauer (1961), Grant (1964) and Lang (1976) among others, widened the field of application of this theory, converting it into a basic tool for the study of surface as well as underground blastings.

A crater blast is that which is carried out with concentrated spherical or cubic charges and with good approximation using relatively short cyllinder charges that are detonated inside the rock mass to be fragmented.

In Fig. 24.2, the influence of the energy transmitted by the explosive to the rock, depending upon the depth of the charge and the volume of material affected by the blast. When the charge has a very shallow burial (a) most of the energy is transmitted to the atmosphere in form of airblast, up to an excessive depth (c) where all the energy is applied upon the rock, fragmenting it and producing a high intensity vibration. Between the two situations, there will be one that produces a larger crater.

In the cavities formed, three different concentric zones can be found: the aparent crater, the true crater and the mound of fragmented rock, Fig. 24.3.

The mound is subdivided into the zone of complete fragmentation and that of extreme or tensile fragmentation. In blastings with inverted faces, ths crater sizes are influenced by the effect of gravity and the structural characteristics of the rock, forming elongated, elliptic shaped cavities which correspond to the extreme rupture or stressed zones, Fig. 24.4.

The basic parameters for crater blasting are:

- The ratio Length/Diameter of the cylindrical explosive charges should not exceed 6 to 1 so that they perform as spherical charges.
- The burial depths, distance between the center of gravity and the free face, should be the optimum which is determined through practice applying the Livingston theory.
- The drilling pattern is calculated from the optimum depth and maximum volume of the craters.

Livingston determined that a relationship existed between the critical depth D_c, from which the first signs of external action in the form of cracks and fractures are noted, and the weight of the explosive Q, in agreement with the empirical equation:

$$D_c = E_t \times Q^{1/3} \text{ (Equation of Strain-Energy)}$$

where: E_t = Strain-Energy Factor, which is a characteristic constant in each Rock-Explosive combination.

The preceding equation can be written in the following manner:

$$D_g = \Delta \times E_t \times Q^{1/3}$$

having: D_g = Distance from the surface to the center of gravity of the charge, Δ = Relationship of depths, a dimensionless number equal to D_g/D_c.

The burial depth, where the explosive maximizes the crater volume V is known as the optimum depth D_o, therefore:

$$\Delta_o = D_o/D_C$$

where: Δ_o = Optimum depth relationship.

In order to determine the optimum burial depth, a series of tests will be carried out with attention to the following recommendations:

- The tests will be done on the same type of rock and

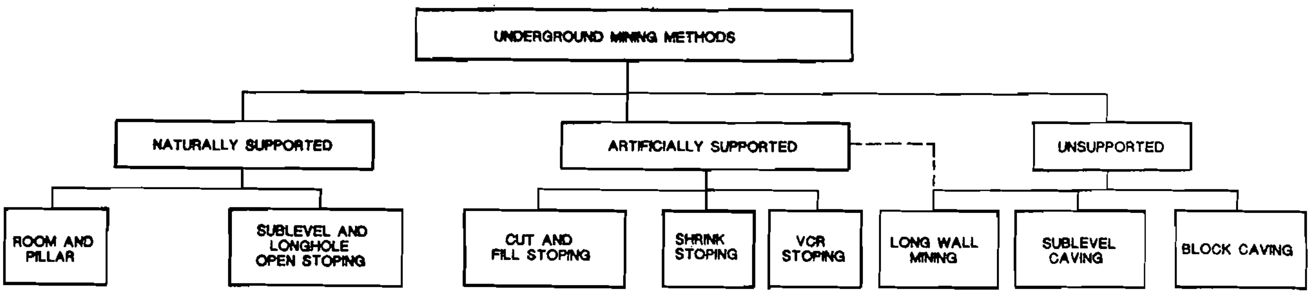
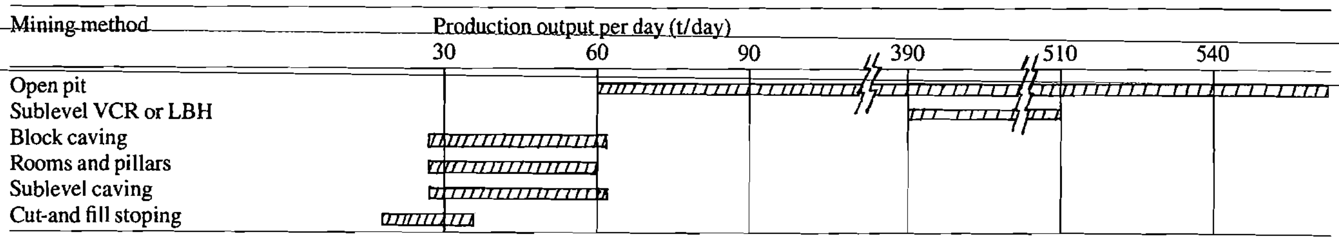


Fig. 24.1. Underground mining methods.

Table 24.1.



with the same explosive that is to be used in the production blasts.

– The blasthole diameters will be as large as possible, for example 115 mm.

– The series of pilot hole lengths will be as ample as possible to allow an ample range of burial depths, for example 15 blastholes between 0.75 and 4 m with increases of 0.25 m.

– The blastholes will be placed perpendicularly to the free face.

– The explosive charges will have a length of 6 D and will be adequately stemmed.

After each test, the volume of the crater will be measured, and afterwards, with all information in hand, the Volume-Depth curve will be established, Fig. 24.5.

In order to better describe the rock breakage procedure and the importance of charge shape, Livingston also proposed the following empirical equation:

$$V/Q = E_t^3 \times A' \times B' \times C'$$

(Equation of the Fragmentation Process)

where: A' = Coefficient of the use of explosive energy, B' = Coefficient of the behavior of the material, C' = Coefficient that takes into account the effects of the geometry of the charge.

If the charges used are spherical and the depth is optimum, the value of B' can be determined by the preceding equations because $A' = C' = 1$, $V = V_o$, and therefore:

$$B' = V/D_c^3$$

As in this type of blastings it is necessary to maximize the efective energy developed per unity of charge length, the explosives used will comply with the following characteristics: high detonation velocity, high density and the possibility of completely filling the cross section of the blasthole.

The ideal explosives for hard rock are the watergels, emulsions and gelatin dynamite, and in medium and soft rocks the lower density and velocity watergels. ANFO has a very limited range of application and is only used in soft rocks.

24.2.2 *Mining method with vertical crater retreat 'VCR' stoping*

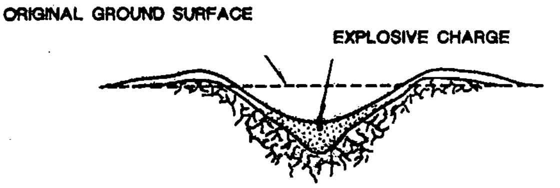
This method consists basically in delimiting the ore stope that is to be exploited by a system of shafts directed to a different level, by drilling from the charging level the whole series of blastholes that cover the room and firing them in successive upward rounds with elongated sperical charges $L < 6$ D, placed in the optimum depth in such a way that the excavated craters overlap to form as regular a roof as possible, Fig. 24.6.

The broken ore is hauled away through cross cuts excavated from the extraction level to the draw point of the stope. The extraction is usually done in a controlled manner, evacuating only the quantity of ore necessary to allow sufficient space between it and the stope ceiling for the next round, avoiding detachments from the side walls which would dilute the ore.

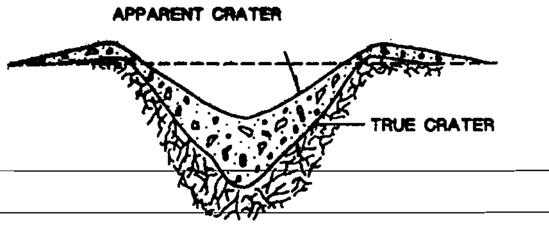
Once the deviations of the drills are under control, as well as the height of the cut in each of the craters excavated in each round, procede to charge the explosive, after plugging the lower part of the blastholes by one of the systems shown in Fig. 24.7.

Once the explosive charge has been placed at the adequate depth with its initiator and/or multiplicator, it will be stemmed to improve the confinement with a length of inert material 12 times the diameter of the blasthole, using fine sand or water to avoid the risk of obstruction, Fig. 24.8.

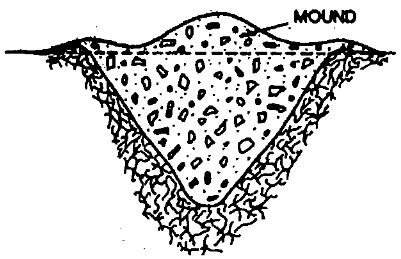
In this type of work a determined initiation sequence is not necessary, as in bench blasting, owing to the characteristics of the breakage mechanism in crater blasts. However, when there are charges under the mean level of



a. SHALLOW BURIAL ~ 0,2m/Kg$^{1/3}$



b. OPTIMUM APPARENT CRATER ~ 0,6m/Kg$^{1/3}$



c. MOUNDING DEPTH ~ 0,1m/Kg$^{1/3}$

Fig. 24.2. Effects of increasing depth of burial on crater shapes.

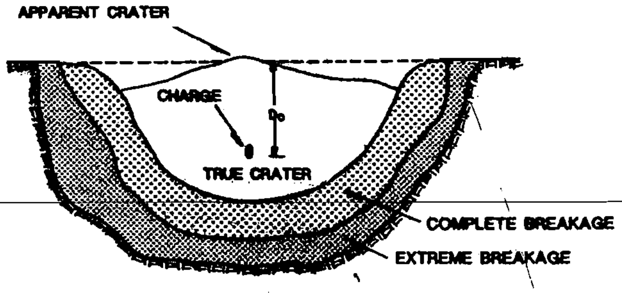


Fig. 24.3. Zones of a crater.

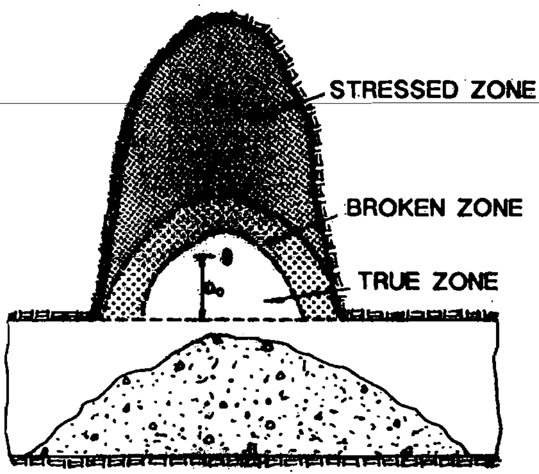


Fig. 24.4. Dimensions of the cavities created by spherical charges with inverted faces.

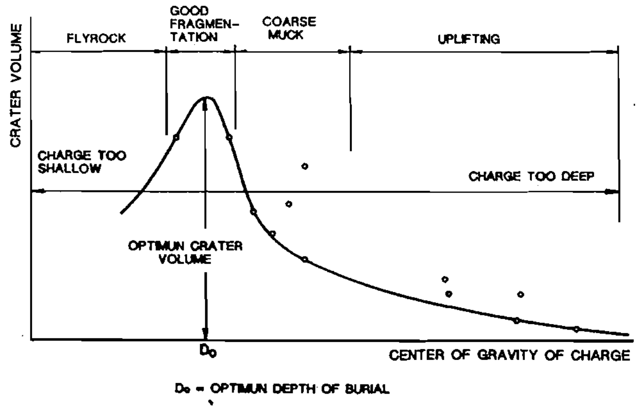


Fig. 24.5. Demonstration of the results of crater blastings.

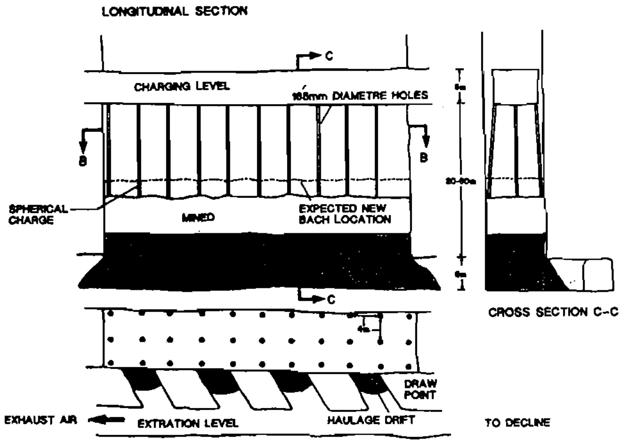


Fig. 24.6. Pattern of the 'VCR' stoping mining method.

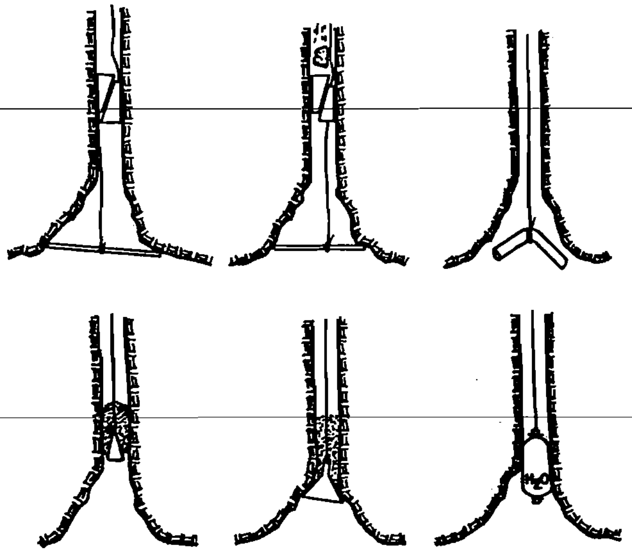


Fig. 24.7. Different ways of plugging holes (Mitchell).

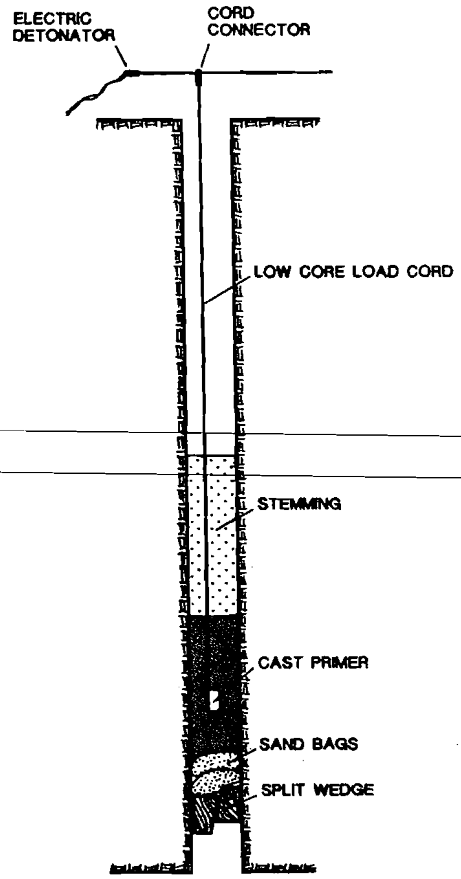


Fig. 24.8. Design of the charge in a blasthole.

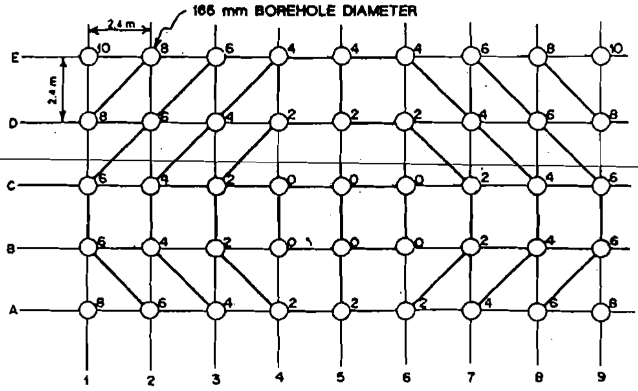


Fig. 24.9. Typical vertical crater retreat blasting delay pattern.

the stope ceiling it is recommended that they be fired first. It is also convenient, whenever possible, that each charge have two free faces, thus increasing fragmentation. In the pattern of Fig. 24.9, a typical sequence in this method is representated, so that the blastholes of the same number have two free faces, one is the stope ceiling and the other the walls of the craters excavated in earlier blasts.

The breakage of the final crown of the pillar, which is right underneath the charging level or topsill, requires the use of special blasts which can be designed knowing the mean vertical advance in each round and the dimensions of the crown. As a general guide, the criteria in Table 24.2 can be followed.

24.2.3 *Advantages and inconveniences of the 'VCR' method*

This method has the following advantages:

- Optimum safety for personnel and equipment (excluding the last blast in which the crown is broken).
- With warehouse chambers, the protection needed for the orebody side walls is reduced as the broken and swollen ore acts as support.
- As the charge weights per hole or delay are small, the vibration levels are not very high.
- The fragmentation is usually good.
- Muck loading, without remote control, can reach 70% and, if there is lateral access, it can be up to 80%.
- It is well adapted to narrow orebodies of around 3 to 10 m thick, even with inclinations that are not very elevated, and
- No need to drill raises.

Photo 24.1. Test crater blast.

Table 24.2.

Dimension of crown*	Blasting procedure
< 1.5 X	Single blast. Single symmetrically placed charge
1.6-2.0 X	Single blast. Decked charges fired simultaneously
> 2.0 X	Two separate blasts

*Function of the mean vertical advance per round X.

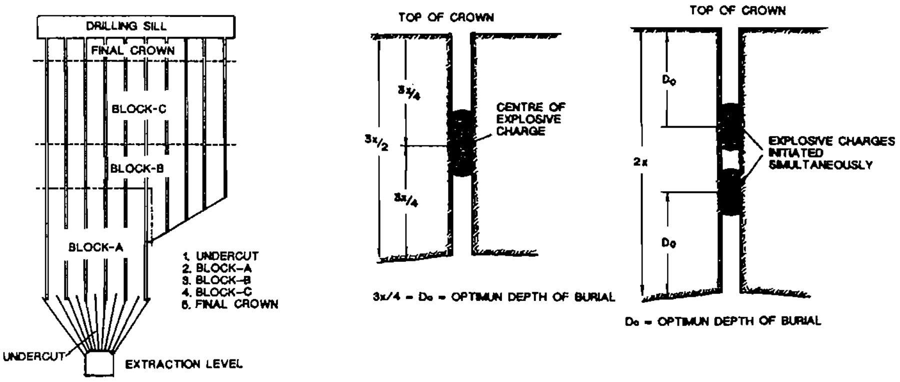


Fig. 24.10. Design of a crown blast (Hagan).

On the other hand, the problems are:

– Poor ventilation while loading the ore, requiring secondary ventilation.

– The damage to hanging walls is extensive with risk, on occasions, of caving.

– Grade control is difficult because in each round the muck piles up on that of the preceding blast and mixes during its descent, and

– After the extraction, rock can fall off the side walls and increase dilution.

24.3 LONGHOLE METHOD

24.3.1 *Long blasthole mining method 'LBH'*

The longhole method, 'LBH – Large Blasthole', is an application of the principals of open pit bench blasting to underground mining. The method affects primarily the breakage operation and, in a certain way, the preparation of stopes as, in general, the work is carried out on two sublevels, one for drilling and another for extracting. However, the operational methods are the same as in the conventional sublevel open stopping, Fig 24.11.

In the 'LBH' method, each stope is divided into three clearly differentiated sectors:

– The undercut, which carries out the mission of receiving the fragmented ore and creating a free face in the bottom of the blastholes.

– The longhole sector, where the large diameter pilot holes are drilled and represent between 85 to 90% of the chamber tonnage.

– The slot raise, which is used as the first vertical free face of the blast, for the undercut as well as for the longhole zone, Fig. 24.12.

The slot raise, or beginning of the sector, is constructed from a raise with dimensions that oscillate between 1.8 and 3.5 m, depending upon the case, and which can be excavated with a raise borer or by the 'VCR' method using the available drilling equipment.

From the raise, the undercut is created with vertical fan shaped holes, usually of 65 mm, on a pattern of 1.5 x 2 m in the bottom of the blastholes. The powder factor is around 800 g/t.

Afterwards, with the production drilling equipment, large diameter blastholes are opened (165 mm) on a triangular pattern, Fig. 24.13.

24.3.2 *Blasting in the longhole method 'LBH'*

To calculate the drilling pattern in the long blasthole zone, Langefors equation is usually applied:

$$B_{\max} = \frac{D}{33}\sqrt{\frac{\rho_e \times \mathrm{PRP}}{\bar{c} \times f \times (S/B)}}$$

where: $B_{\max}$ = Maximum burden (m), D = Blasthole diameter (mm), $\bar{c}$ = Rock constant (Taking generally: $\bar{c}$ = 0.3 + 0.75 Medium hard rocks; $\bar{c}$ = 0.4 + 0.75 Hard rocks), f = Fixation factor (Vertical blastholes; f = 1, Inclined blastholes; 3:1 f = 0.9; Inclined blastholes 2:1 f = 0.85), S/B = Relationship between spacing and burden, ρ_c = Charge density (kg/dm^3), PRP = Relative weight strength of the explosive.

The value of the practical burden is established from the maximum value, applying a correction for the deviation of the blastholes and collaring errors:

$$B = B_{\max} - 2D - 0.02\,L$$

where L is the blasthole length.

The spacing S is determined with the equation:

$$S = 1.25\,B$$

The drilling pattern influences the sizing of the drifts or cuts of the top level of drilling.

The bench blasting in this method does not require toe breakage, therefore it is only necessary to use the column charge. The most used explosives are: ANFO for hard and medium hard rock, and ALANFO for very hard rock. If there is water present in the blastholes, the charge can be placed in a plastic covering or use watergels and low density emulsions.

The main problem in this type of blastings is the level of vibrations generated by the large quantity of explosive that can be accomodated in the blastholes.

These vibrations produce dynamic stresses that can cause damage in the underground labors or the nearby installations.

This problem is solved by decking the charges with intermediate stemming or wooden separators. After studying the vibrations, the maximum quantity of explosives needed to form an elemental charge should be determined by observing the following:

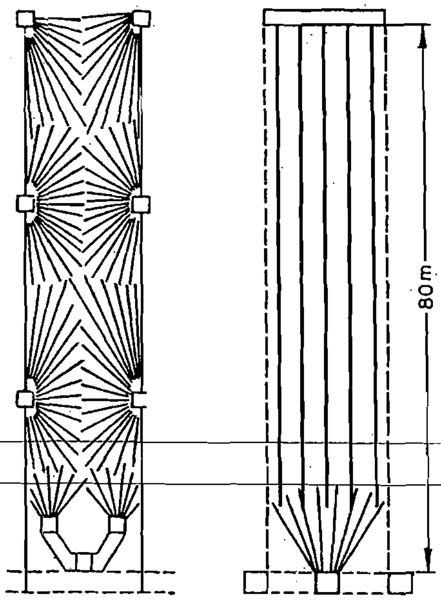
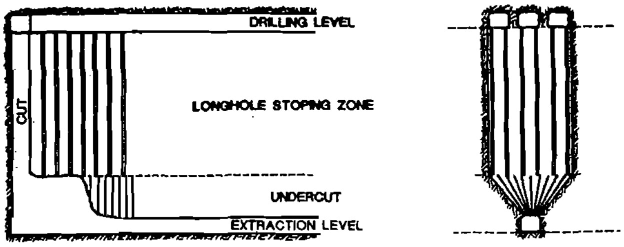


Fig. 24.11. Blasthole drilling patterns for conventional sublevel open stoping and for long blastholes.



a) LONGITUDINAL SECTION

b) TRANSVERSAL SECTION

Fig. 24.12. Representative sectors of the long blasthole method.

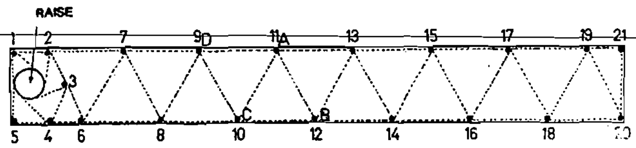


Fig. 24.13. Slot blasting to a bored raise with 165 mm blastholes (Hagan).

– The relationship Length of charge/Diameter should be maintained above 20 to obtain a good fragmentation.

– The volume of rock in front of the intermediate stemmings tends to make fragmentation worse.

– A large subdivision of the blastholes makes for a more complex charging operation and initiation system.

The length and type of intermediate stemming between deck charges should be such as to:

– Not produce sympathetic detonation or desensitivation of the adjacent charges that are initiated at different times.

– Achieve adequate fragmentation along the length of the stemming column.

– Assure that the stemming material has a size distribution close to 1/20 *D*, to inhibit secondary sulphate blasts, if necessary.

Therefore, the criteria for sizing should be, according to the type of explosive, Table 24.3.

The elemental charges by delay oscilate between 100 and 200 kg and are carried out as shown in Fig. 24.14.

The initiation of the charges can be done with electric detonators inside the blastholes or with a non-electric system, such as Nonel detanotors, low core load cord with temporized boosters, etc. In all cases boosters are necessary and it is recommended that two per charge be used.

The advantages of the non-electric initiation methods are the following:

– Lower risk of accident produced by premature firing of the detonators.

– Reduced charging time.

– Easy to handle.

– A sequential blasting machine is not necessary.

The initiation sequence of the blast is from bottom up, with the following recommended delay timings. According to Du Pont:

– Charges in the same blasthole . . . . 50 ms.
– Adjacent charges in the same row . . . . . . 10 ms.
– Charges between rows . . . . 75 ms.

Table 24.3.

Explosive	Intermediate stemming
ANFO	12 to 15 D
Watergels	16 to 18 D

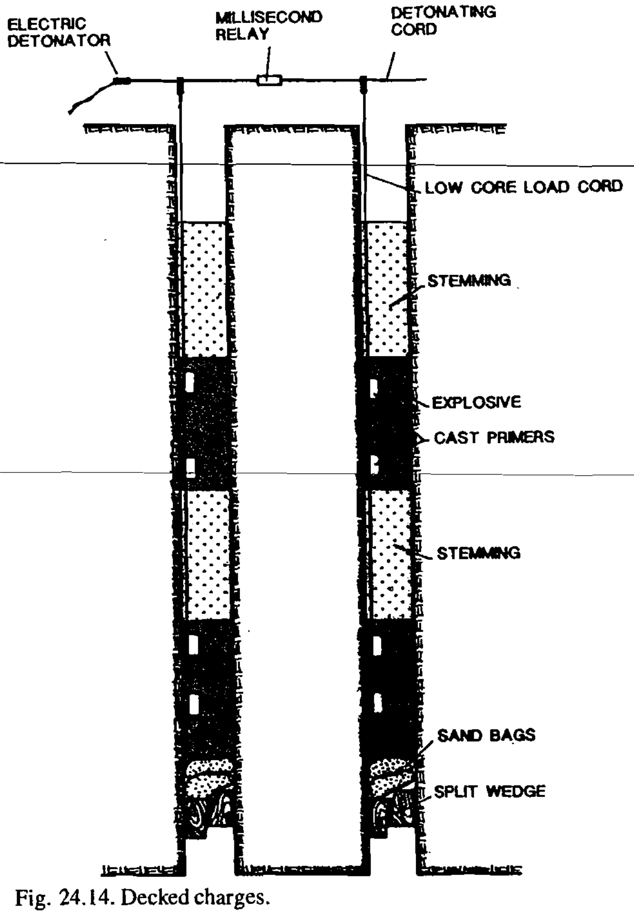


Fig. 24.14. Decked charges.

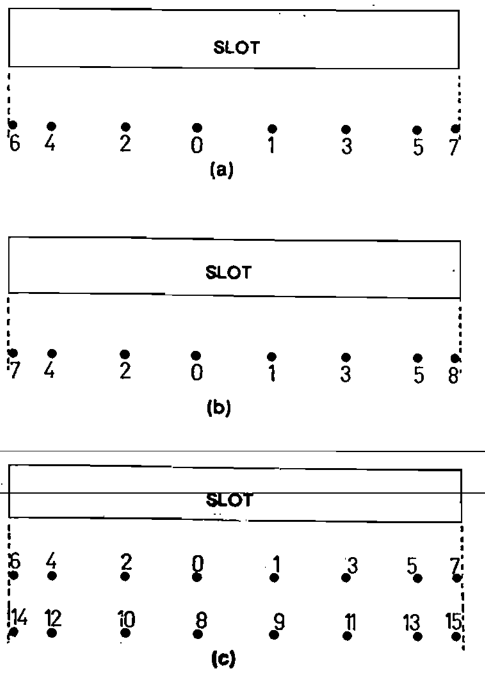


Fig. 24.15. Initiation system for blasts with continuous charges per delay: (a) standard blast, (b) extra relief for pillar-line holes and (c) double-row blast (Hagan).

When firing one row blasts with continuous column charges, the sequence will be such that initiation starts in the center of the row, Fig. 24.15a. When the probability of cut offs is minimal, due to the delay elements being inside the holes, pillar damage can be reduced by setting a long delay timing in the blastholes at each end, Fig. 24.15b.

In double-row blasts, in which each hole contains a continous column charge, the placing of only one delay per charge makes the initiation system more difficult and increases the risk of cut offs owing to the amount of delays existing between a hole in the first row and the adjacent one in the next row, Fig. 24.15c.

24.3.3 *Advantages and inconveniencies of the Longhole method 'LBH'*

The main advantages of the longhole method are:

- Extra safe working conditions and regularity in production.
- High productivity and breakage yield per lineal meter drilled.
- Very high benches, up to 70 m, which make large size blasts possible.
- Less damage to the remaining rock, as the blast has two free faces and the blastholes can be designed with decked charges.
- The possibility of loading up to 80% of the broken rock without remote control.
- Lower powder factor than with the 'VCR' method.
- Use of less expensive explosives such as ANFO instead of watergels or emulsions.
- Lower drilling and blasting costs, and
- Good grade control and low ore dilution.

The main disadvantage of this method is that it causes a compression of the muck because it falls from a great height.

24.4 SUBLEVEL STOPING WITH BLASTHOLES IN FAN PATTERN

This system is applicable to sub-vertical orebeds with side walls that have good characteristics, so that the when the ore is extracted large open stopes remain, similar to those in the 'VCR' and 'LBH' methods.

The drilling is done from drifts on the sublevels in fan shape with upward or downward blastholes, or both, whose lengths are adapted to the surrounding ore. In an effort to reduce the preparation work, which is costly, the blastholes are made as long as possible, Fig. 24.16.

The drilling rigs used are of special design with extension drill steels and bits of 51 to 64 mm. The separation between drilled sections is usually around 1.2 and 1.8 m.

The collaring, orientation and deviation of the blastholes are some of the operative conditions necessary to obtain good blasting results. This means that it is necessary to use special orientation systems and accessories, and not drill blastholes of more than 25 m.

The blasts are done with one free face, and partially removing the muck from the preceding rounds.

The yield of broken rock per lineal meter drilled is low because the planned spacing diminishes as it nears the collar, leaving part of the blastholes unused in the breakage.

The calculations of the drilling patterns are taken from the necessary powder factor, which is a function of the rock type, drilling length and width of the blast.

$$CE = CE_o + 0.03\,L + \frac{0.40}{AV}$$

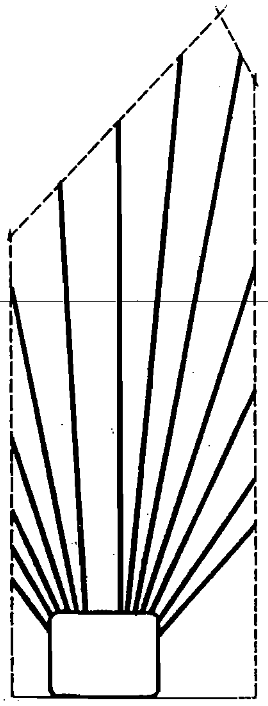


Fig. 24.16. Sublevel stoping with fan holes.

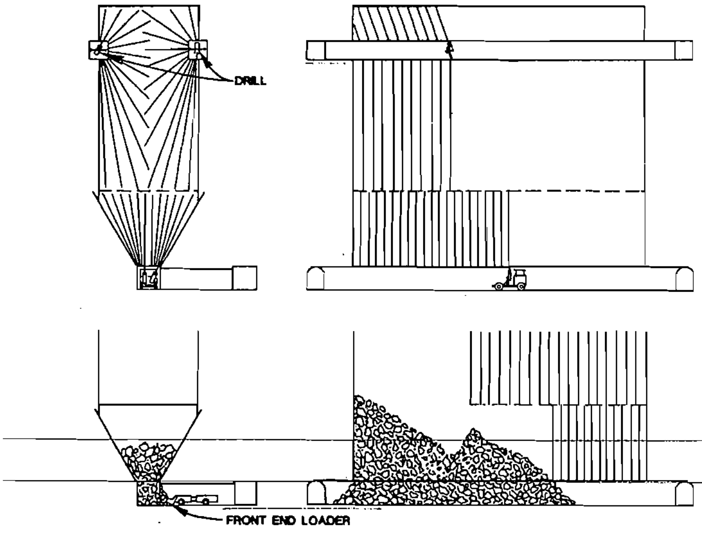


Fig. 24.17. Detail of a drilling pattern.

Table 24.4.

Type of rock	Base powder factor CE_o (kg/m^3)
Fissured and hard	0.6
With joints	0.55
Fractured	0.5
Relatively homogeneous	0.45
Homogeneous and hard	0.40
Soft and homogeneous	0.35

where: CE = Designed powder factor in the bottom of the blasthole and in one fifth the length of the same, expressed in kg/m^3 of gelatin explosive, CE_o = Base powder factor of the rock, calculated from Table 24.4, L = Blasthole length (m), AV = Width of the round (m).

The pattern in the bottom is calculated from the lineal charge concentration q_l that is to be reached, making

$$A_e = S \times B = \frac{q_l\,(\text{kg/m})}{CE\,(\text{kg/m}^3)}$$

proving the ratio:

$$\frac{S}{B} = 1.3 \text{ to } 2$$

having: S = Spacing (m), B = Burden (m).

When $S = 2B^2$ good results are usually obtained and then $A_e = 2B^2$, where the value of the burden is found in order to calculate the spacing.

The column charge is designed between 50 and 75% of the bottom charge, with sufficient length to obtain good fragmentation.

To lower drilling costs, it is necessary to get maximum results from the same based upon mechanized systems of charging.

The most frequently used explosives and blasthole filling equipment are the following:

- Cartridge gels with pneumatic chargers.

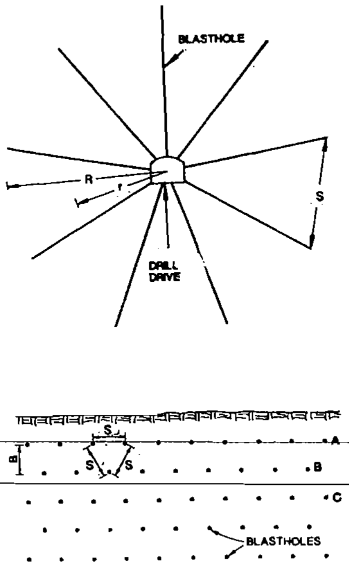


Fig. 24.18. Blastholes drilled in fan pattern in parallel planes following the standard equilateral triangle design (Hagan).

- Water gels and cartridge gels with pneumatic chargers.
- Water gels and bulk emulsions with pump units.
- Bulk ANFO with pneumatic chargers.

Recently, Hagan suggested drilling holes in a fan pattern following reorientated equilateral triangles. In Fig. 24.18, the conventional drilling pattern can be observed.

These patterns bring the following problems:

- The distribution of the explosive energy is only optimum within a cylinder of rock with r radius. As this magnitude diminishes, so does the effectiveness of the pattern.
- When the sector that has been drilled in fan pattern has an angle that is smaller than 360°, Fig. 24.19, the distribution of the energy at the farthest points of the pattern (e.g. in the volume ABC of Fig. 24.18) is unsufficient and, consequently, so is fragmentation and swell.

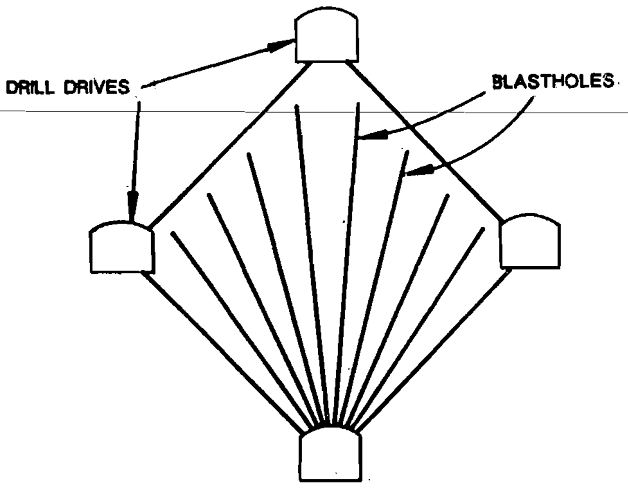


Fig. 24.19. Drilling sector with blastholes in fan pattern with a central angle that is under 360° (Hagan).

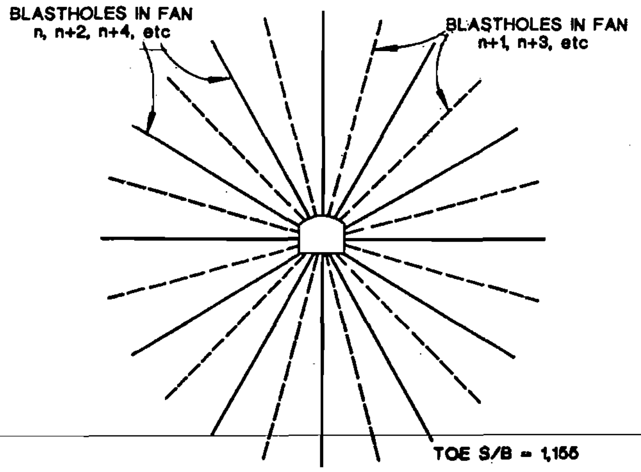

BLASTHOLES IN FAN n, n+2, n+4, etc
BLASTHOLES IN FAN n+1, n+3, etc
TOE S/B = 1,155

Fig. 24.20. Parallel fan holes drilled in a conventional pattern and with the *S/B* relationship value equal to 1.155 at collaring points (Hagan).

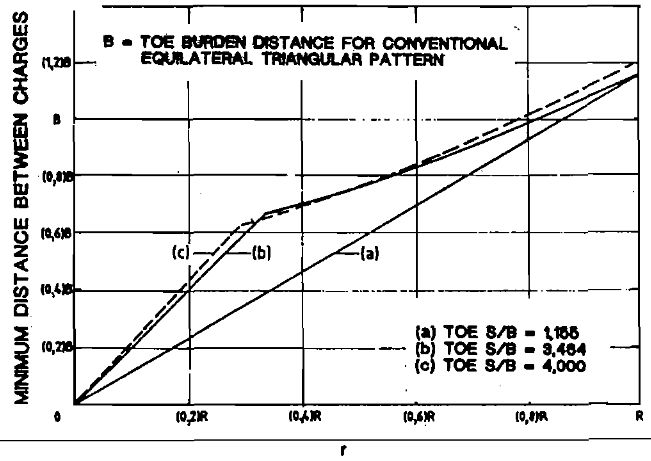


Fig. 24.23. Minimum distance between charges for different drilling patterns and values of *r* (Hagan).

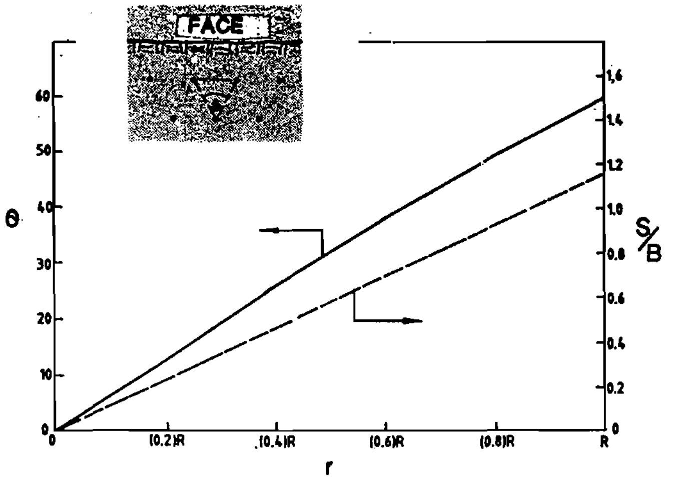


Fig. 24.21. Values of θ and *r* for different *S/B* relationships at collaring points (Hagan).

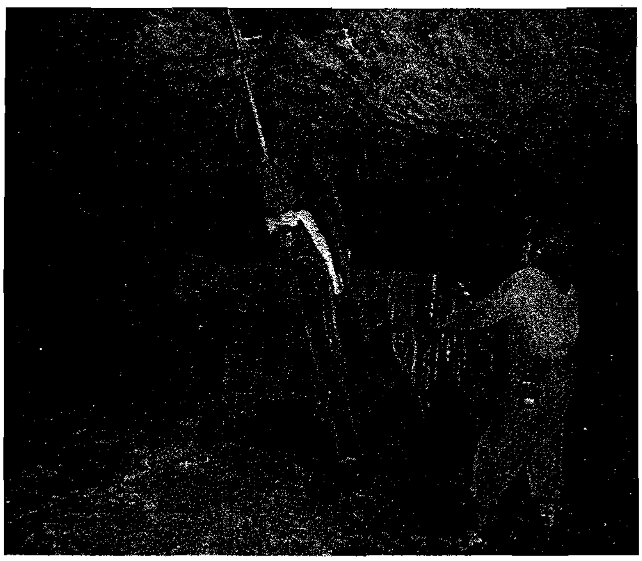


Photo 24.2. Drilling an uphole.

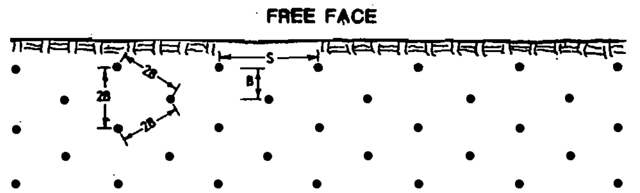


Fig. 24.22. Drilling blastholes in a fan pattern with reorientated equilateral triangle designs (Hagan).

– As the distance between charges in a fan pattern decreases, the probability of a charge initiating, desensitizing or 'robbing. the burden of other adjacent charges increases.

In Fig. 24.20, it can be seen how the parallel fans are dephased from one another with a pattern of equilateral triangles, obtaining a *S/B* relationship value that is equal to 1.155 at the collars.

In Fig. 24.21, one can observe how the values of *S/B* and θ diminish as *r* decreases from *R* to 0. This decrease signifies a smaller amount of energy freed, an increase between the predicted and actual detonation times, more energy lost in vibrations and fly rock, and a poorer distribution of the useful energy.

By rotating the equilateral triangles 30° with respect to the face line, the drilling pattern shown in Fig. 24.22 is achieved.

With these patterns, *S/B* values that are equal to 3.464 can be reached, giving the following advantages:

– For any value of *r* the minimum distance between charges is larger, for an *S/B* relationship at the collaring point of 3.464, Fig. 24.23. For this reason, the probability of the detonation of a charge initiating or harming another adjacent one with a later break time is smaller. For the reorientated patterns, the minimum distance between charges refers to between adjacent fans if $R > r > 0.3$ or within the same fan, as happens in the conventional patterns, if $r < 0.3R$. When *S/B* is under 1.0, the capacity of a charge to initiate or harm an adjacent charge rapidly increases.

– When the *S/B* relationship at the collaring point is over 2.4, part of the ore is fragmented by sections of charges near the collar that break towards balanced concave biplanar faces. As *S/B* increases from 2.4 to 4.0. the percentage of fragmented ore increases from 0% up to 64%. This only happens in reorientated patterns.

– As *r* diminishes, the blastholes deviate from their initial reorientated pattern, but they go through a conventional equilateral triangle pattern upon a cylindrical surface that has a radius of $r = 0.3\,R$.

Therefore, it can be deduced that the reorientated patterns with collar *S/B* relationships close to 3.5 allow a better use of the explosive energy, with the consequent reduction in drilling and blasting costs, avoiding the potential problems of unstability, and producing lower vibration levels.

24.5 ROOM AND PILLAR MINING

Used in horizontal or flat dip bedded deposits of salts, limestone, potash, iron, etc, with maximum inclination of 30° and rocks with stable geomechanic characteristics, ore that can be extracted by excavating large chambers and leaving pillars to support the side walls.

The pillars are placed in regular patterns, usually square, circular and sometimes rectangular. Their sizing is one of the most important aspects because it conditions the ore recovery and the stability of the operation.

In horizontal deposits, or with little inclination, the process consists in opening drifts for the extraction and haulage of the ore. This is often done by connecting these labors with the previous drifts. The dimensions of the excavations correspond with the height and width of the drifts, and the machinery used is composed of jumbos with several booms and drilling rigs which gives a high degree of mechanization.

The inclined orebodies are divided vertically into levels, from which the haulage drifts are established along the foot walls. These drifts serve as access ways to the production areas where the operation continues upwards to the next level. The drilling for the blasts in these areas is carried out with hammers installed upon pushers. Mechanization is not wide spread in this method, which means abundant manual labor and low productivity.

A variation of the room and pillar method in inclined orebeds is composed of sloping drifts which serve as access to the operation zones and can be climbed by the jumbos. From these access drifts, other horizontal drifts are excavated at intervals, following the ore boundary as closely as possible.

In thick deposits, the excavations cannot be carried out with the jumbos in one phase, which means that the ore is divided vertically, and the lower part is recovered by bench blasting. The benching is carried out with conventional rigs and vertical blastholes, Fig. 24.24.

In these mining operations, roof bolting is a technique that is applied extensively to improve stability which, apart from production drilling, should be taken into consideration as a corresponding part of the work.

24.6 CUT AND FILL MINING

This method was originally developed in Canada in the later fifties. It consists in excavating the ore by ascending horizontal sublevels, filling the holes produced by ore extraction from the deepest sublevels with waste material, which serves as support for the hanging walls and as a work platform for the drilling, charging and haulage equipment. Hydraulic filling is a usual practice because of easy transportation and the possibility of mixing it with a small percentage of cement, Fig. 24.25.

This method has the following advantages:

- High ore recovery.
- Grade and dilution control.
- Mechanization of the operations.
- Fewer problems of side wall stability and surface caving, and
- Easy and effective ventilation.

Two systems can be used for drilling and blasting:

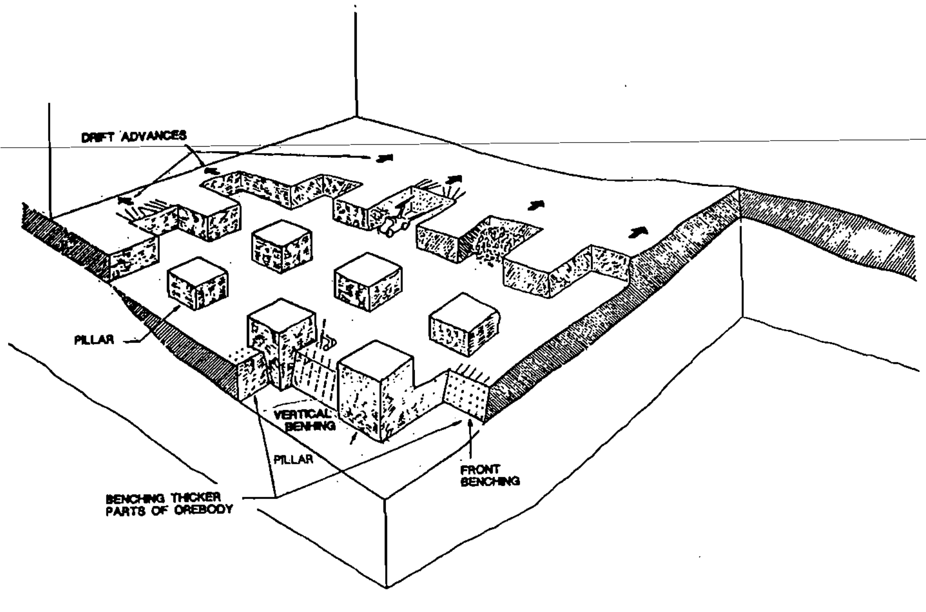


Fig. 24.24. Room and pillar mining.

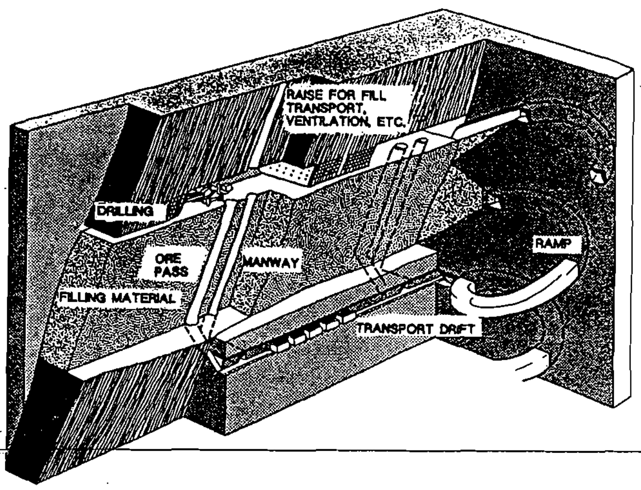


Fig. 24.25. Cut and fill mining method.

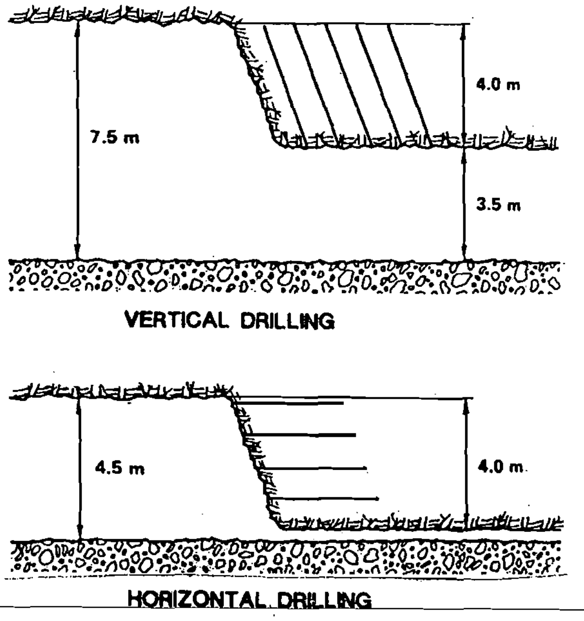


Fig. 24.26. Systems of drilling and blasting.

vertical drilling into the roof by the inverted bench method, and horizontal drilling, as in a bench that is turned 90°, Fig. 24.26.

The first system has the advantage of being able to drill a large area quite in advance of the blasting and charging operations. The quantity of ore extracted can be as large or small as desired. The drilling does not usually surpass four meters vertically, because the total height of the excavation would be excessive and the blastholes are placed with a 50 to 65° inclination. One inconvenience of the system is the irregularity and poor condition of the roof after each blasting. In some mines it is necessary to previously reinforce the ore mass with cemented cables to avoid its caving in.

The second system consists in drilling horizontal blastholes with the same length as the drill rods, also less than the four meters, and shoot towards the lower free surface. The volume of each blast is limited because it has to be drilled from the face. The advantages of this alternative system are: a more regular roof surface, lower height of the hole opened, improved selectivity and ore recovery.

Although hand held drills and pushers can be used for the drilling with blastholes of 29 to 33 mm diameters, the normal practice is to use drilling wagons and jumbos with larger diameters such as 33 to 64 mm.

The explosives used go from the conventional and the slurries to ANFO, with powder factors that vary between 200 and 260 g/t.

When drilling is vertical, the stemming of the blastholes is done with clay plugs.

24.7 UNDERGROUND CHAMBERS IN CIVIL ENGINEERING PROJECTS

During the last decades there has been an increase in the use of underground space for diverse projects:

- Hydroelectric power stations.
- Fuel storage.
- Depots for toxic and radioactive residues.
- Atomic shelters.
- Military instalations.
- Underground parking lots.

In relation with the height of the chambers, these are classified as follows:

- Small, with a total height of under 10 m.
- Large, of up to 60 m or more.

24.7.1 *Small chamber stoping*

This type of chamber is similar to large section tunnels with the excavations carried out in the same manner. Generally, the mining is divided into phases which begin in the top part with a pilot drift (1) and lateral stope holes (2), to follow by benching in the lower part with horizontal or vertical blastholes (3), Fig. 24.27.

So as to not harm the rock and diminish the thickness of the concrete lining, smooth wall blasts will be carried out in the final profiles.

The lower bench is broken once the upper section is finished, or with a certain time difference. The stoping can be done with horizontal blastholes, using the same jumbo of the other sections, or with vertical blastholes, using a surface drilling rig. The normal drilling diameters for the horizontal holes are from 32 to 45 mm and 50 mm for the vertical holes.

24.7.2 *Large chamber stoping*

In large chambers, the normal practice is to carry out the excavations from top to bottom in descending phases, with different drilling and blasting systems, Fig. 24.28.

Phases 1 and 2 are done with horizontal drilling and removal of broken rock through the pilot drift. The phases 3, 4 and 5 are normally carried out with vertical drilling and rock removal through the different access levels which, in the case of the hydroelectric power stations, can be the alternators, lower tracts of pipelines, drainage drifts, etc.

The vertical blastholes are usually limited to 4 m in

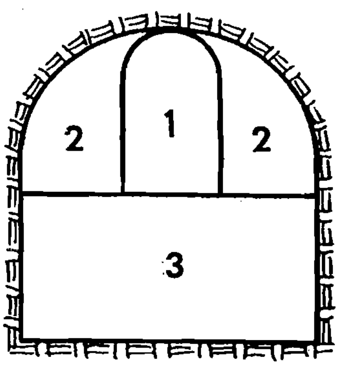


Fig. 24.27. Stoping phases in a small chamber.

height, because they permit the use of a *glory hole* raise for the evacuation of the broken rock.

The excavation procedures vary with the quality of the rock, but in any case it is done with the idea of assuring contour stability as it advances downwards. The excavation of the top part or arch is carried out in phases, as indicated, from a drift that is used for examination. If the ground is good, a systematic bolting of the surface with gunite reinforcement steel ribs is usually performed and, if the rock is poor, the excavation is carried out in transversal sections, leaving between them others that are not excavated until the arches already opened have been concreted.

In the hydroelectric power stations, the end of the top section depends upon the method chosen for support of the guide rails for the crane bridges. The most difficult to solve is when they rest on the actual rock. In this case, underneath the top section level a trench cut is performed to later carry out two cross cuts, in two horizontal planes, and two presplits, usually in the vertical planes. This way, the corners of the rock are formed, assuring their integrity and resistance.

Bench blasting is the easiest and therefore should be used as much as possible. The bench heights are not usually very large because the blasts are done in relatively high conditions of confinement, drifts usually enter in the central cave and it is necessary to support the side walls as the excavation procedes downwards. For these reasons, the horizontal and vertical drilling should be

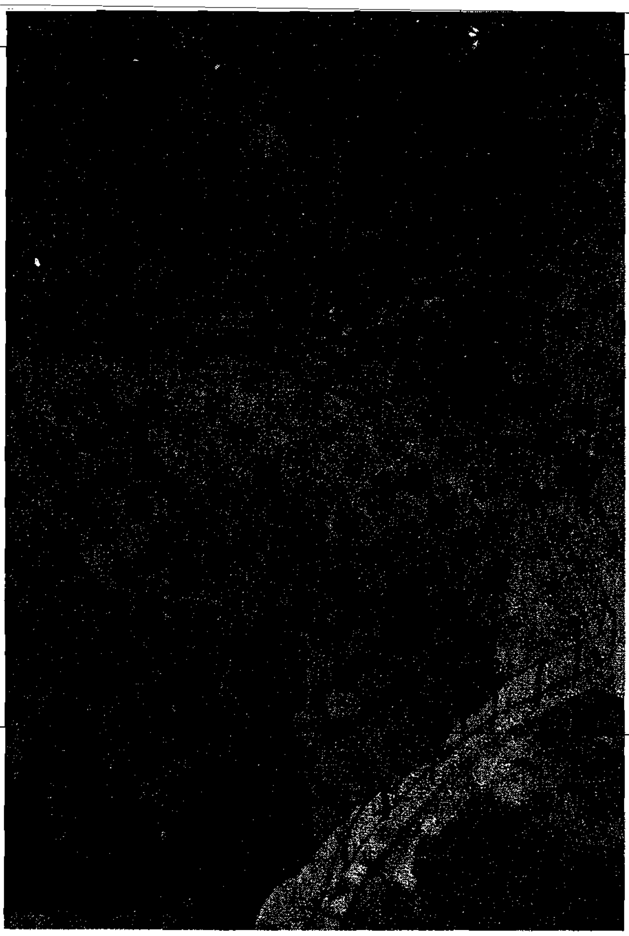


Photo 24.3. Hydroelectric Power Station of Saucelle (Iberduero, S.A.).

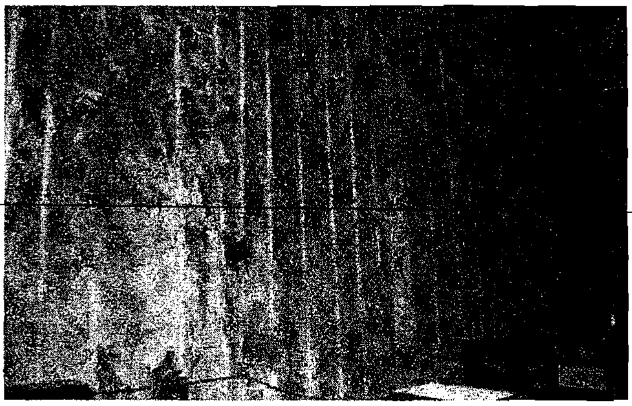


Photo 24.4. Side walls with presplitting in the Hydroelectric Power Station of Saucelle (Iberduero, S.A.).

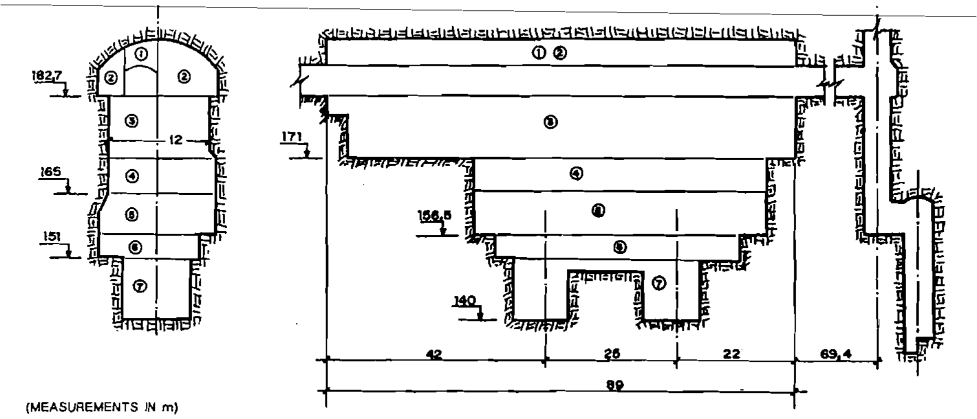


Fig. 24.28. Excavation phases in Aldeadávila II (Courtesy of Iberduero, S.A.).

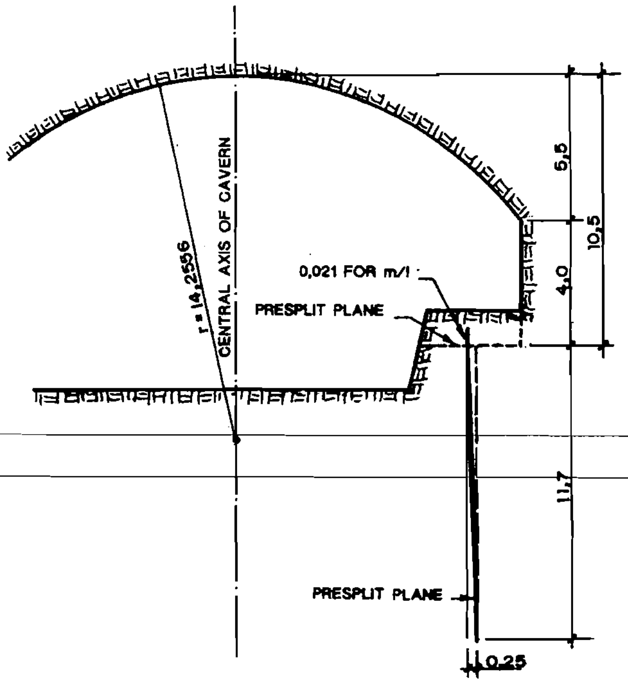


Fig. 24.29. Detail of chamber with support of guide rails.

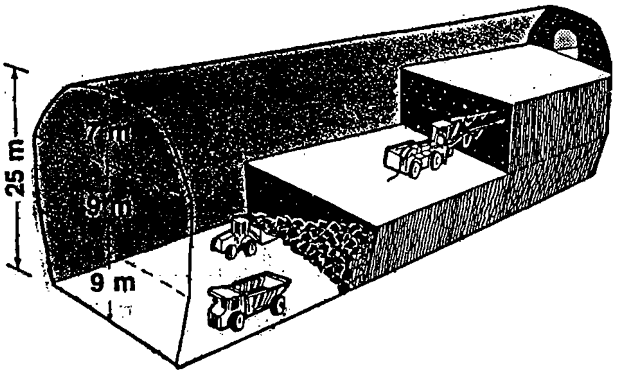


Fig. 24.30. Chamber excavation with two benches and horizontal driling with a jumbo.

simultaneous, apart from the availability of different drilling rigs.

For the removal of the broken rock from the lower parts of the hydroelectric power stations, it is recommended that the accesses be drilled beforehand, at the same time that work is being done in the top part of the caves.

As for the calibers of the blastholes, the usual ranges are 32 to 45 mm for the horizontal drilling, and from 50 to 65 mm for the vertical.

REFERENCES

Atlas Copco: *Manual Atlas Copco* 4th edition, 1984.

Botin, J.A.: *Método VCR y de Barrenos Largos en Minería* I Seminario de Ingeniería de Arranque con Explosivos en Proyectos Subterráneos. Fundación Gómez Pardo. 1986.

Bryce, R.C.: *Blasthole stoping and filling in weak ground.* CIM Meeting, 1985.

Crocker, C.S.: *Vertical crater retreat mining at the Centennial Mine.* MCJ, June, 1979.

Espa, A.: *La Implantación del Método de Cráteres Invertidos en la Mina de Almadén.* J.M.M., Húelva, 1980.

Fora, J.: *Excavaciones de Cámaras en Centrales Subterráneas.* I Seminario de Ingeniería de Arranque con Explosivos en Progectos Subterráneos. Fundación Gómez-Pardo, Marzo, 1986.

Guilfoyle, K.S. & N.L. Bradford: *Production blasting at Isa Mine.* Underground Operators Conference. AIMM, 1982.

Gustafsson, R.: *Técnica Sueca de Voladuras.* SPI, 1977.

Gustavsson, K.: *Sublevel stoping in the Stiassa Mine.* First Int. Symp. on Rock Fragmentation by Blasting. 1983.

Hagan, T.N.: *Optimizing the yield and distribution of effective explosion energy in fans and rings of blastholes.* The AusIMM, Explosives in Mining Workshop, Melbourne, Victoria, 1987.

Hermanson, L.: *Production drilling with high accuracy.* First Int. Symp. on Rock Fragmentation by Blasting. 1983.

Harries, G. & T.N. Hagan: *Some considerations of blasting with large diameter blasthole underground.* A.M.F., 1979.

Julia, J.E. & F.S. Rojas: *Explotación con Tiros Largos de Gran Diámetro en la Mina El Soldado-Chile.* Revista Minerales, vol. 39.

Jorgerson, G.K.: *A review of Vertical Crater Retreat Mining.* M:C:J: July, 1981.

Kelly, J.G. et al.: *Surface crown pillar blast at Shervitt's Fox Mine.* 1986.

Lafrance, R.: *Application of large-diameter boreholes in underground mines.* CIM, Bulletin, January, 1984.

Lang, L.C.: *The application of spherical charge technology in stope and pillar mining.* E/MJ, May, 1976.

Lang, L.C.: *Spherical charges development vertical crater retreat method in stope and pillar mining.* SEE, 1978.

Lang, L.C.: *Vertical crater retreat: An important new mining method.* Underground Mining Methods Handbook, AIME, 1982.

Lopez Jimeno, E. & C. Lopez Jimeno: *Las Voladuras en Cráter y su Aplicación a la Minería.* VII Congreso Internacional de Minería y Metalúrgia, Barcelona, 1984.

Lopez Jimeno, C. & C. Lopez Jimeno: *El Seccionado de Cargas: Una Técnica Eficaz para Disminuir el Nivel de Vibraciones en Voladuras.* Canteras y Explotaciones, Noviembre, 1986.

Maclachlan et al.: *Spherical charge cratering-plane and angle geometry involved in small scale single and row tests.* CIM Bulletin, May, 1981.

Mitchell, S.T.: *Vertical crater retreat stoping as applied at the Homestake Mine.* Homestake Mining Co, 1980.

Mitchell, S.T.: *Vertical crater retreat stoping proves successful at Homestake Mine.* ME, November, 1981.

Neindorf, L.B. & P. Stevens: *Current drilling practice at Isa Mine.* Underground Operators Conference, AIMM, 1982.

Niklasson, B.: *Besok Vid Nagra Australiensiska Gruvor Som Tillàmpar Kratersprångnings Metoden.* SveDeFo, 1982.

Page, C.M.: *An adaption of big hole blasting underground.* Int. of Mining and Metallurgy, April, 1982.

Quintana, H.: *Grandes Excavaciones Subterráneas en Minería.* II Seminario de Ingeniería de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, 1987.

Redpath, B.B.: *Application of Cratering Characteristics to Conventional Design.* Monograph on Rock Mechanics Applications in Mining, 1977.

Sanmartin, V.C.: *Diseño de Grandes Voladuras.* II Curso sobre Control de Vibraciones Producidas por Voladuras, Fundación Gómez-Pardo, 1983.

Singh, K.H.: *Evaluation of blasthole stoping alternatives.* Anaconda Minerals Co., 1985.

CHAPTER 31

Structure and building demolition

31.1 INTRODUCTION

The use of explosives for demolition of civil engineering structures such as buildings, bridges, chimneys, etc., constitutes an economic and sometimes complementary system of the conventional ones that are accomplished by hand or with mechanical means.

The demolition of buildings by controlled blasting is a technique that was developed in Europe during the reconstruction of the cities destroyed in the 2nd World War and, owing to its advantages, it was later extended to the rest of the world.

These labors with explosives consist in placing small charges in strategic points of the structures, provoking unstableness and fragmentation during the fall in a prefixed direction.

In this type of demolitions, the following general principles must be taken into account:

- The breakage of the constructive elements by eliminating unions and sectioning rigid parts so that once instability is induced the structure collapses with the force of gravity and its own weight does most of the demolition.
- Division and distribution of the charges to achieve a complete breakage, maintaining a complete control over flyrock and generated vibrations.
- Proper choice and application of the initiation sequence to provoke the fall of the structure in the desired direction.

The advantages of demolitions with explosives over the classical systems can be summed up in:

- Lower global cost, especially when the structures have elevated heights.
- More rapid execution.
- Safer when carried out on or near traffic arteries.
- Disturbances limited to a short time.
- High work control.
- Possibility of using conventional loading procedures.

On the other hand, some of the inconveniences are:

- A project and complete study of the blast is necessary.
- The loss of time dedicated to obtaining permits and legal transactions.
- The impossibility to recuperate some constructed elements of value.
- The necessity of interrupting the traffic in the vicinity of the blast.
- Sometimes there are no building plans of the structures and the characteristics of the building materials are not known.
- The safety measures that should be taken during the work are:
- The explosive charges must be covered with adequate protection such as heavy splinter-proof covering material, sandbags, nets, etc., to avoid flyrock.
- Before and during the blast, the structure and area surrounding should be sprayed with water to eliminate dust.
- The surrounding area of the blast should be inspected and evacuated.
- If there are buildings nearby, ground vibration measurements should be carried out.
- If contact charges are to be used, which is not common practice, flyrock and airblast must be controlled.

31.2 DRILLING DIAMETERS AND TYPES OF EXPLOSIVES

As already indicated, in demolitions with explosives, the charges are individual and very small, usually under 50 g and the blastholes are usually situated in points of difficult access. For this reason, the most widely used drills are hammers with a drill diameter of 38 mm.

The equipment weighs 25 kg when vertical holes are drilled with lengths of up to 3.6 m, and light hammers of around 12 kg for horizontal blastholes with a maximum length of 1 m.

Only in the case of large foundations and structures out in the open, where there is no risk of flyrock, is it possible to use drilling rigs with a range of diameters between 50 and 65 mm.

As to explosives, because the greater part of the elements to be fragmented are made up of concrete, bricks, and rarely, of stone, high tension energy explosives are the most adequate, meaning those which have high densities and detonation velocities, along with great utilization sensitivity and safety. Therefore, the gelatins in cartridges of 22 and 26 mm diameter are the most widely used.

Other explosives which can be frequently used for this type of work are : the powders, the watergels and the high grammage detonating cords.

In the following, the powder factors refer to gelatin explosives with a density in cartridge of 1.4 g/cm^3, and a detonation velocity of about 5.200 m/s.

Photo 31.1. Preparation of the explosive charge.

Table 31.1.

Material	Powder factor CE (kg/m^3)	Drilling B = S (m)
Non-reinforced concrete of poor quality	0.25-0.30	0.70-0.80
Non-reinforced concrete of good quality and material strength	0.30-0.40	0.60-0.70
Reinforced concrete with only surface reinforcement	0.60-0.75	0.50-0.60
Reinforced concrete with heavy reinforcement	0.80-1.0	0.50-0.55
Extra powerful reinforced concrete of military type	1.5-2.0	0.40-0.50

Table 31.2.

Thickness of wall H_m (cm)	Pattern – B X S (cm)	Number of rows
35	30 × 30	2
45	35 × 35	2
60	45 × 45	2
70	55 × 55	3
> 100	55 × 55	3

Table 31.3.

Thickness of wall H_m (cm)	Pattern	Number of rows
35	25 × 25	2
45	30 × 30	2
60	45 × 45	3
70	50 × 50	3

31.3 DEMOLITION OF STRUCTURAL ELEMENTS

In the design of the blasts, a first division is made between isolated structural elements and complete structures or construction.

In the first group the following elements are taken into consideration:

- Foundations
- Concrete or cement walls
- Pillars
- Slabs
- Roofs, and
- Beams.

31.3.1 *Foundations*

The foundations are drilled with vertical blastholes and with lengths that depend upon its size and with a square pattern $B = S$.

The explosive charges are prepared by uniting pieces of about 50 g to a detonating cord and with a spacing that is in function with the density of the calculated charge.

In Table 31.1, the specific charges and patterns recommended by Gustafsson in function with the material of the foundation are indicated.

In order to obtain a good fragmentation which facilitates loading, at the same time that the flyrock is controlled, an initiation sequence with millisecond delay detonators is recommended.

31.3.2 *Walls*

The following types of walls are seen:

- Brick walls that are part of the reinforcing elements of the structure.
- Concrete walls that pertain to the sustaining structure.
- Concrete walls, cast into the ground but not subject to stress.

Brick walls

In order to save drilling time, this should be carried out at the height of existing spaces, such as doors and windows, to achieve greater breakage and easier drilling.

The number of rows should never be less than 2 and should be increased if the wall is to collapse.

The recommended patterns in function with the thickness of the wall are indicated in Table 31.2.

Depending upon the quality of the material, consistency and risk of flyrock, the specific charge oscillates between 0.5 and 1.0 kg/m^3.

Drilling lengths depend upon the thickness of the wall H_m. Under normal conditions, the holes are drilled with $L = \frac{2}{3}H_m$ leaving a stemming of $T = \frac{1}{3}H_m$ and a charging zone of the same dimensions, Fig. 31.1.

The placing of the blastholes can be done on a staggered square pattern which allows better distribution of the explosive.

Concrete walls

In the case of reinforced concrete walls that form part of the retaining structure, the patterns used are detailed in Table 31.3.

As in the previous case, the hole lengths will be $\frac{2}{3}$ the thickness of the wall, but the powder factors will increase up to 0.9-1.5 kg/m^3, Fig. 31.2.

Concrete walls cast into the ground

In these instances, where the walls are usually high, narrow and cast into the ground, the blastholes are drilled vertically to achieve fragmentation of the concrete and facilitate its digging and haulage, Fig. 31.3. It is not recommended that the holes be longer than 1.5 m, to avoid deviations which could cause flyrock problems due to the existance of high charge concentrations.

In Table 31.4, the design parameters of the blastings in function with the sizes and construction materials of the walls are indicated.

The initiation should be carried out with millisecond delay detonators and the surfaces that are to be blasted should be carefully covered.

In this type of blastings, small experimental tests should be made on part of the wall to determine the ideal patterns and charges.

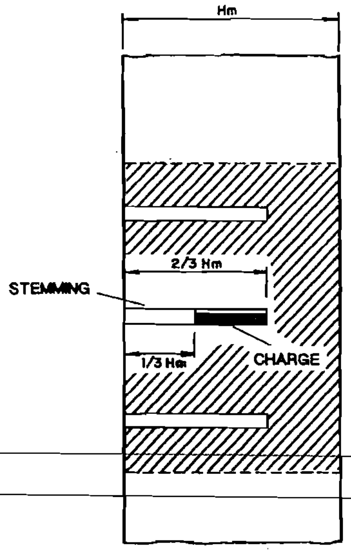


Fig. 31.1. Transversal section of a wall and the geometry of the charges.

Table 31.4.

Type of wall	Thickness H_m (cm)	Spacing S (cm)	Rows	Powder factor CE (kg/m^3)
Non-reinforced concrete walls	20	30	1	0.2-0.3
	30	30	1	0.2-0.3
	40	30	1	0.2-0.3
	50	40	1	0.2-0.3
Reinforced concrete walls	20	30	1	0.3-0.5
	30	30	1	0.3-0.5
	40	30	1	0.3-0.5
	50	40	1	0.3-0.5
	60	40	2	0.3-0.5
	70	40	2	0.3-0.5

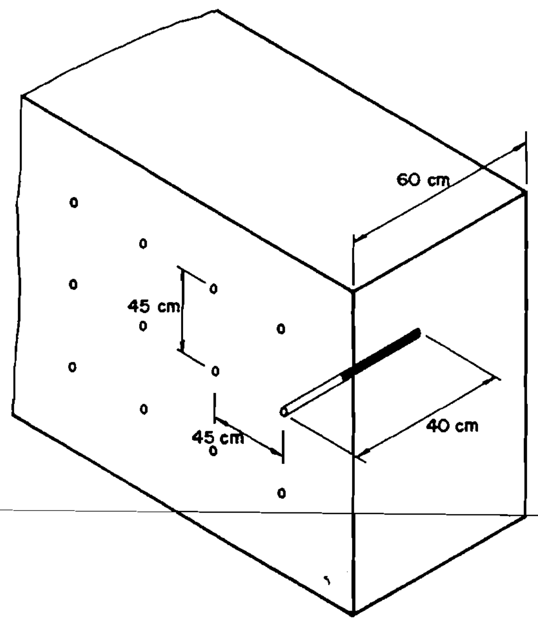


Fig. 31.2. Placing of the blastholes in a concrete wall.

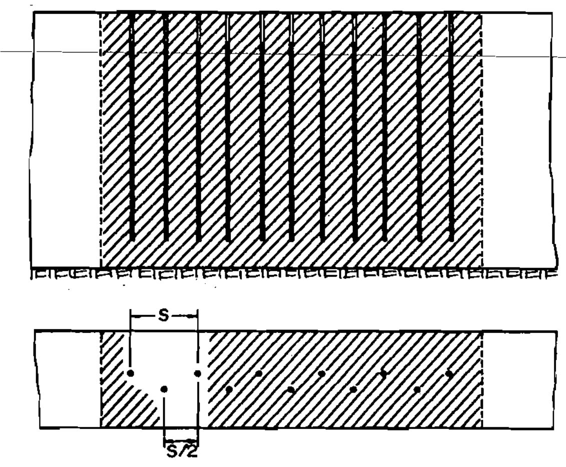


Fig. 31.3. Volume of a high and narrow wall.

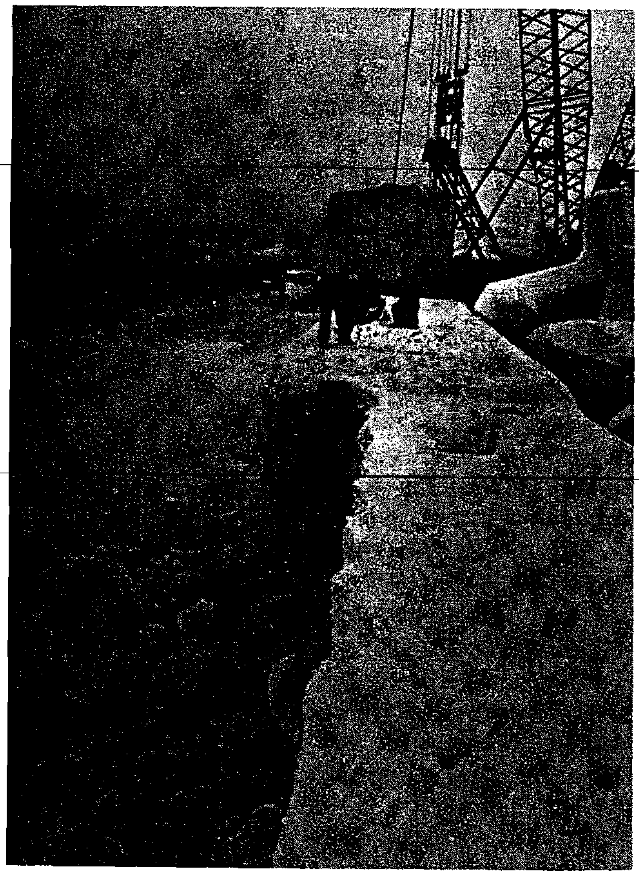


Photo 31.2. Partial demolition of a breakwater wall.

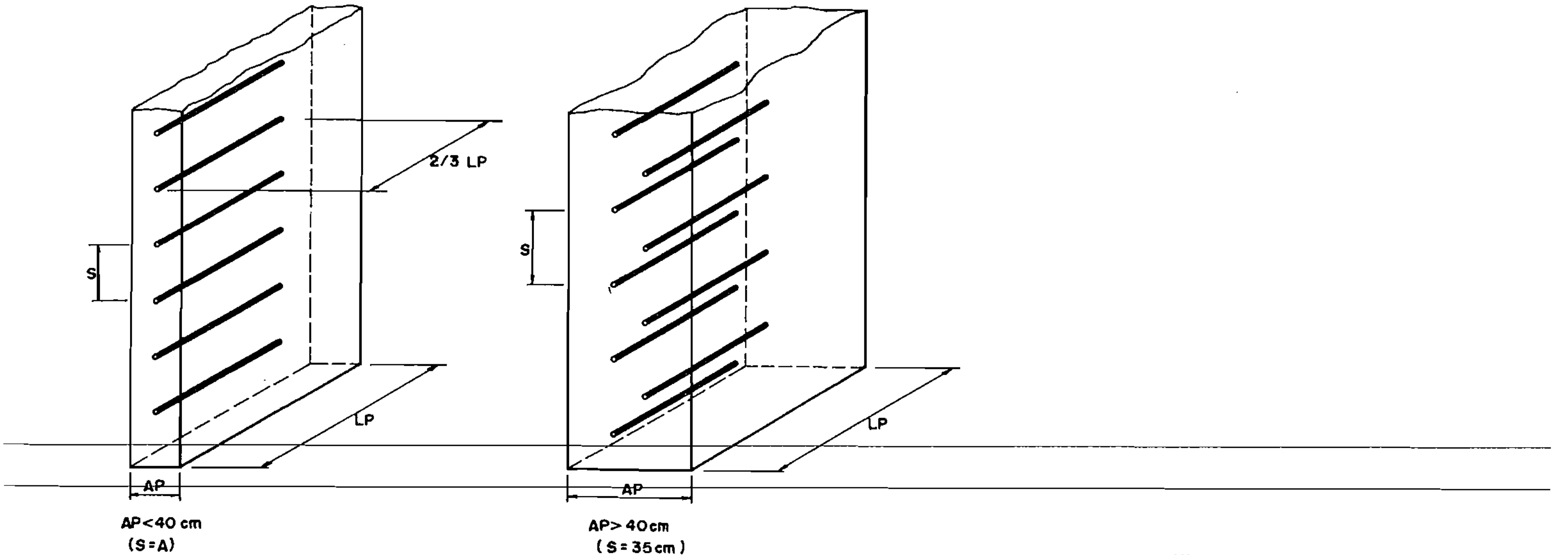


Fig. 31.4. Concrete pillar blasting.

Table 31.5.

Material	Thickness (cm)	Drilling length (cm)	Pattern B × S (cm)	Powder factor CE (kg/m³)
Non-reinforced concrete	30	20	30 × 30	0.3-0.5
	40	25	40 × 40	0.3-0.5
	50	35	50 × 50	0.3-0.5
Reinforced concrete	30	20	20 × 20	0.5-0.7
	40	30	30 × 30	0.5-0.7
	50	40	35 × 35	0.5-0.7

Table 31.6.

Thickness – H_m	Drilling length (cm)	Pattern B × S (cm)	Powder factor CE (kg/cm³)
20	15	30 × 30	0.5-0.7
30	20	30 × 30	0.5-0.7
40	30	30 × 30	0.5-0.7
50	$\frac{2}{3} \times H_m$	50 × 50	0.5-0.7

31.3.3 *Pillars*

Pillars are usually of reinforced concrete with square, rectangular or circular sections.

The blasthole drilling should be directed parallel to largest face of the pillar.

In pillars with a width under 40 cm, the holes are drilled in a single row with spacing equal to the pillar width, Fig. 31.4. In larger pillars, two rows of staggered blastholes are drilled as in the pattern of Fig. 31.4.

The drilling length should be ⅔ of the largest face size *LP*, and the stemming and charge should occupy lengths of ⅓*LP*.

The height of the cut of the pillars will vary between 1.5 m for those with reduced sections (< 40 cm), to 2.5 m for larger pillars (> 40 cm). Above these zones of principal breakage, other auxiliary ones can be placed every 2 m, with a couple of blastholes each.

The specific charge varies in function with the material and surrounding conditions between 0.7 and 1.5 kg/m³.

31.3.4 *Slabs*

There are two procedures to blast slabs:

- Charging for total fracturation
- Charging to open clefts.

The first case applies to non-reinforced as well as reinforced concrete slabs, while the second method is more frequently used for reinforced concrete where, once the cleft is opened, the steel bands can be cut with a blowpipe, Fig. 31.5.

In Table 31.5, the patterns, drilling lengths and blasthole charges recommended are indicated.

When cleft blastings are to be carried out, at least two rows of blastholes are used, and the powder factor is increased up to 0.7 and 0.8 kg/m³. As for walls, a small test blast should be carried out to adjust charges and patterns.

31.3.5 Roofs

Roofs mean reinforced concrete ceilings and the domes.

In Table 31.6, the design criteria are shown for these blasts.

31.3.6 *Beams*

With the blasting of these structural elements, two objectives are persued:

- Transform the cut point into a hinge joint.
- The fragmentation or plaster shooting of the material for its posterior manipulation.

The drilling is done vertically, with two blastholes per cut point and spaced by 30 cm. The drilling length should be 0.7 times the thickness of the beam and the specific charge between 0.6 and 0.8 kg/m³.

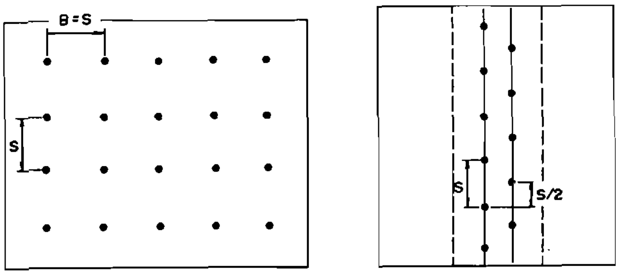


Fig. 31.5. Drilling patterns for slabs.

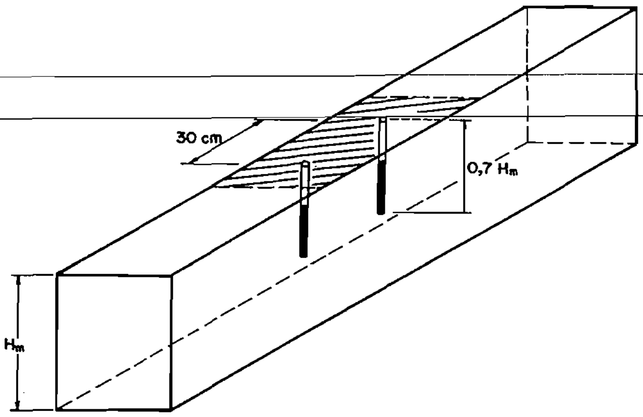


Fig. 31.6. Breakage of a beam.

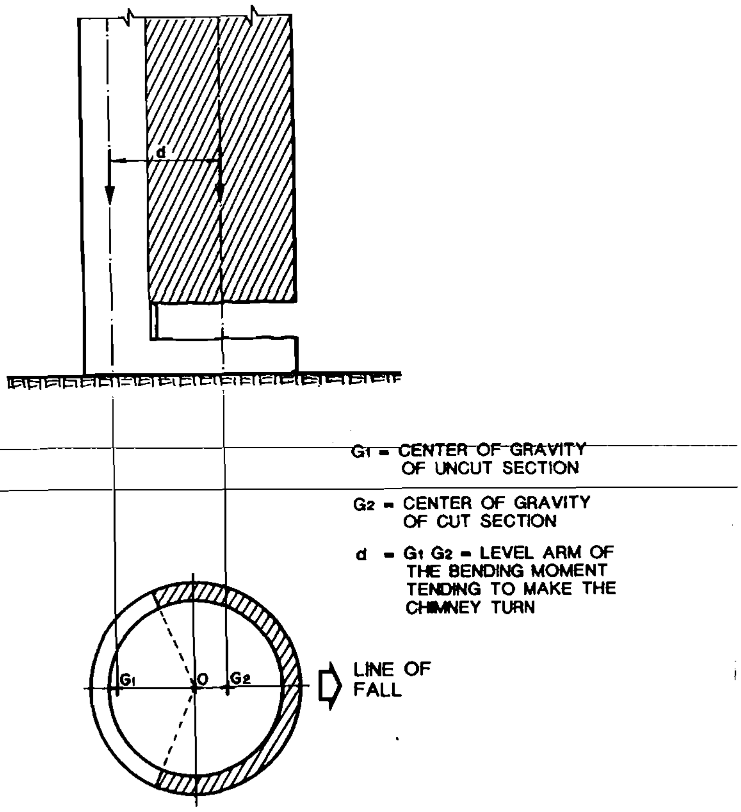

d
G1 = CENTER OF GRAVITY OF UNCUT SECTION
G2 = CENTER OF GRAVITY OF CUT SECTION
d = G1 G2 = LEVEL ARM OF THE BENDING MOMENT TENDING TO MAKE THE CHIMNEY TURN
LINE OF FALL
G1
O
G2

Fig. 31.7. Method of demolishing a chimney stack (Berta, 1985).

31.4 DEMOLITION OF STRUCTURES

The demolition of structures with explosives is one of the most rapid, safe and efficient systems that can be applied at present, but it needs a complete project of the blastings where the design of the same is planned in function with the geometric and tensile strengths of the parts that constitute the operation, the breakage sequence of the retaining elements, the forseen line of fall and direction of collapse, etc.

However, it is not always possible to obtain all the necessary information, because if the constructions are old there are usually no building plans and, of course, the quality of the materials used and their characteristics in points of difficult access are unknown.

Because of all this, a previous examination, analysis and study of the structures to be demolished, based on samples, drilling, partial demolition of the partitions, etc., followed by complementary studies: supports, undermining, unions with cables or bolts, etc., which all contribute in a decisive manner to the success of the demolition.

Briefly, a few typical cases of structures that are demolished with explosives will be studied.

- Chimneys and towers.
- Bridges.
- Buildings of cement and reinforced concrete.

31.4.1 *Chimneys*

The industrial chimneys, or stacks, usually are annexed to other installations, and have circular, square or polygonal sections, constructed of brick or reinforced concrete.

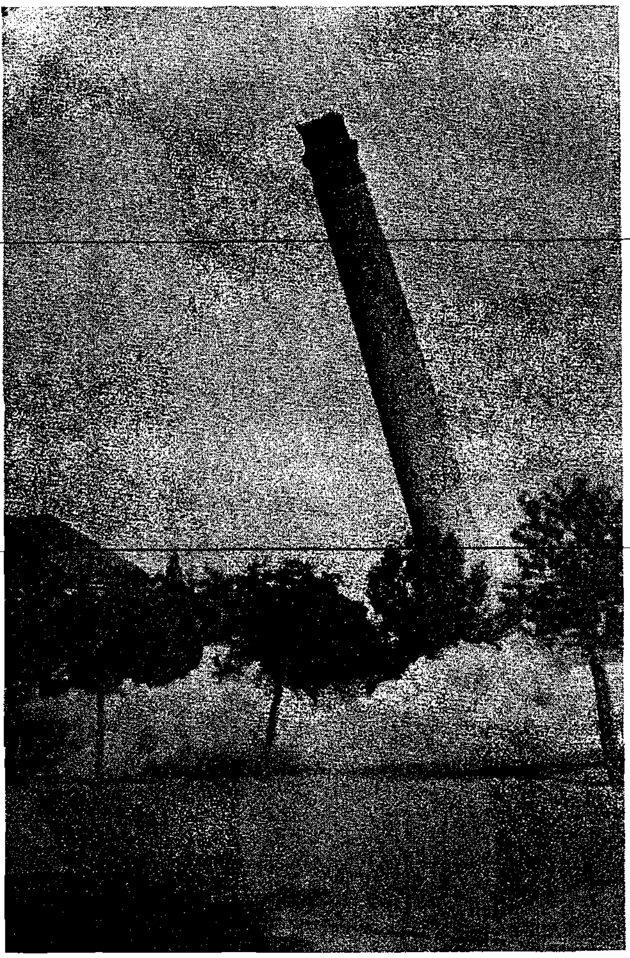


Photo 31.3. Fall of a brick chimney.

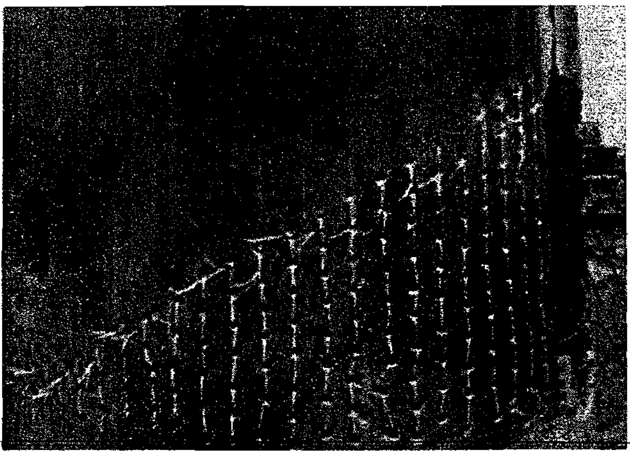


Photo 31.4. Blasting pattern with detonating cord of a reinforced concrete chimney of reduced thickness. (Courtesy of CAVOSA)

Owing to their slenderness, they are the ideal structures to be demolished by explosives. The fall of the chimney is eliminated by blasting part of the sustaining base in such a way that the vertical of the center of gravity moves out of the remaining base, producing a lack of balance of the construction and a turning of the same in a determined direction, as happens during the felling of a tree, Fig. 31.7.

The techniques of chimney demolition can be classified as:

- Directional blasting with complete fall.
- Directional blasting with reduced fall.
- Collapse blasting.

Directional Blasting with complete fall

This is the safest method and the study of its application should be done with priority over the other two methods. Besides, it has the advantage of using less quantity of explosive and fewer blastholes.

To obtain optimum results, the chimney stacks should be in good condition. If they are made of brick, the zone to be undermined should occupy half of the horizontal base on the side of the direction of the fall, which will coincide with the radius that is perpendicular to the diameter and acts as a hinge. The number of blasthole rows necessary is usually three, along the length of ¼ of the perimeter, and two in the rest of the semi-perimeter, Fig. 31.8.

In the chimneys made of reinforced concrete, the zone to be blasted will occupy ⅔ of the base, coinciding the direction of the fall with the bisector of the central angle of the remaining base, which is 120°. The number of rows in the high part of the wedge cut is limited by a height equivalent to ⅓ of the chimney diameter, ending in the lateral parts of the blasted area which occupies ⅔ of the perimeter, in a height equivalent to 3 rows. To achieve better breakage of the material, it is convenient to open some spaces similar to windows with a width that is double the thickness of the chimney and a height slightly above the third row of blastholes.

Directional blasting with reduced fall

In this type of demolition, once the chimney has become slightly inclined, the remaining base is blasted to provoke collapse. This is achieved by firing a row of blastholes on the side opposite the fall, with a delay timing of 0.25 s, with respect to the primary blast, Fig. 31.10.

This method is only applicable in chimneys made of brick.

Collapse blasting

As in the case before, the collapse blastings can only be done in brick chimneys. The rounds consist in the instantaneous firing of a group of charges located in the whole base in an homogeneous manner.

Generally, the number of blasthole rows is two or three, as shown in Fig. 31.11.

This procedure is the most risky and should only be used as an alternative to the aforementioned in an extreme case. The study of the blasting should include meticulous inspection of the condition of the structure to avoid falls in any other radial direction than planned.

31.4.2 *Towers*

Towers are those constructions that have a height of more than 10 m and a base of less than ⅓ of H_t, Figs 31.12 and 31.13.

For the demolition of these structures, the same criteria as for chimneys is followed. In directional blastings with complete fall, the height of the wedge cut will depend upon the thickness of the construction and the width of the base, oscillating between 3 and 20 times the wall thickness.

In towers built upon concrete pillars, the pillars will be blasted following the same procedure and using an initiation sequence with milisecond delay detonators.

31.4.3 *Bridges*

The types of bridges that are apt to be demolished with explosives are:

- Brick or masonry bridges.
- Steel bridges upon masonry supports.
- Concrete bridges.

In general, these labors are characterized by the fact that the debris must be removed very quickly, as there are usually roads or rivers underneath, and because there are usually other nearby structures such as buildings, new bridges, etc.

Below, for each of the types of bridges, general outlines for blasting are given.

Masonry bridges

This is the oldest type of construction and it usually has pillars, abutments and arches of masonry of quite large size, along with materials like mass concrete injected with gravel, Fig. 34.14.

Owing to the robust nature of the sustaining elements, the number of necessary blastholes is elevated.

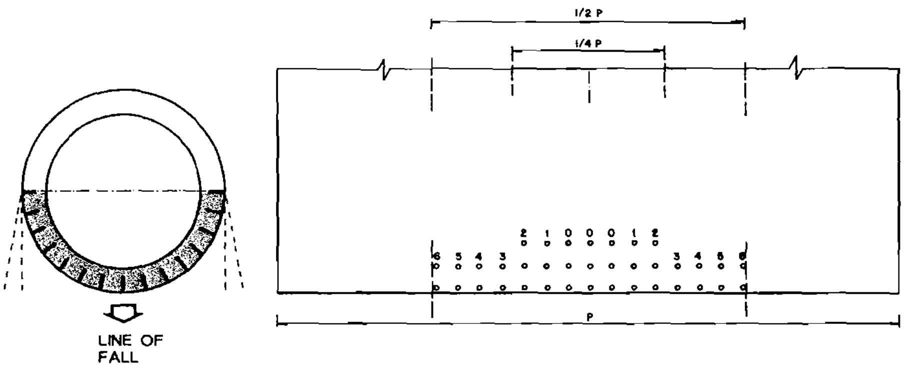


Fig. 31.8. Drilling pattern for a brick chimney.

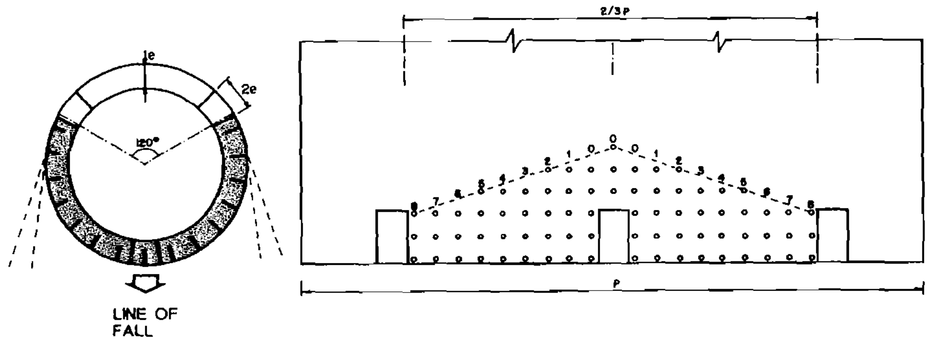


Fig. 31.9. Drilling of a concrete chimney.

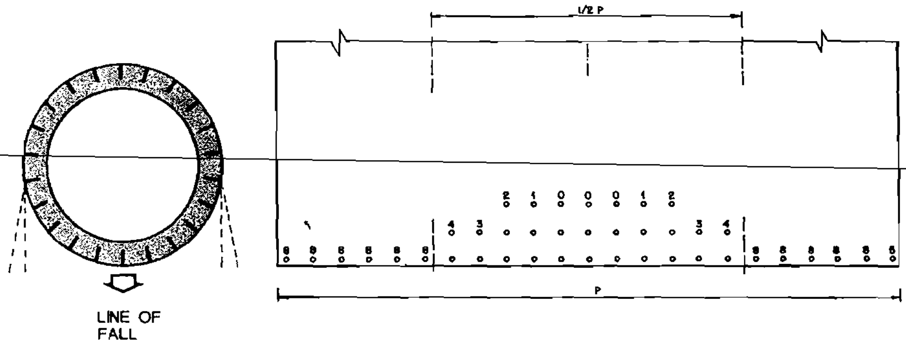


Fig. 31.10. Reduced fall demolition of a chimney.

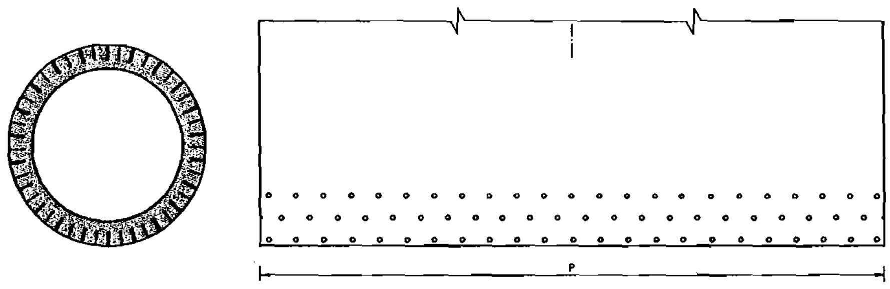


Fig. 31.11. Collapse demolition of a chimney.

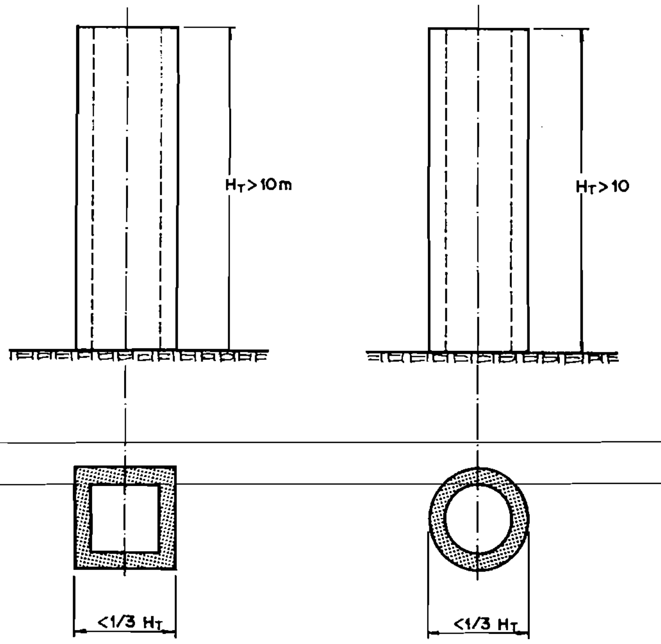


Fig. 31.12. Types of towers.

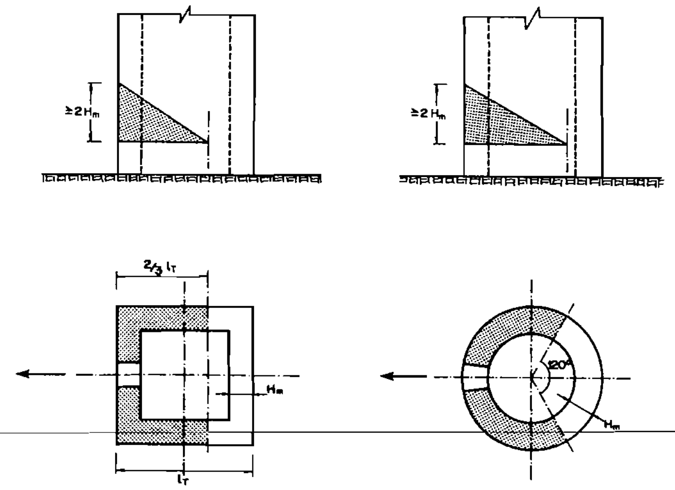


Fig. 31.13. Demolition of square or circular section towers.

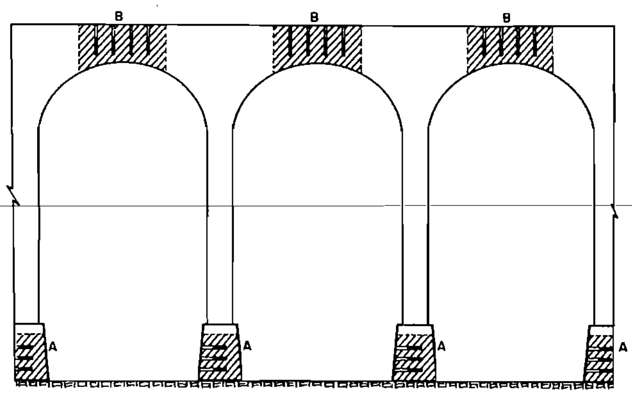


Fig. 31.14. Blasting zones in a bridge of masonry.

Whenever the surrounding conditions allow, the drilling should be done with rigs instead of by hand, opening holes of 50 mm in diameter.

The critical zones of the demolition are the abutments and the arches. The first, as far as drilling patterns and charging goes, as brick walls, while in the arches, according to their width, the blasting can be global or in sections as indicated in Fig. 31.15.

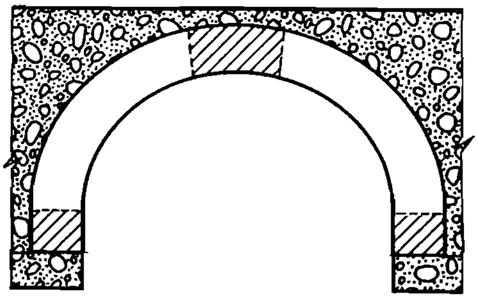


Fig. 31.15. Blasting of an arch in sections.

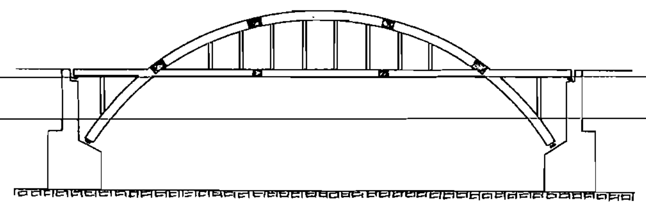


Fig. 31.16. Blasting zones for the demolition of a arch beam bridge (Berta, 1985).

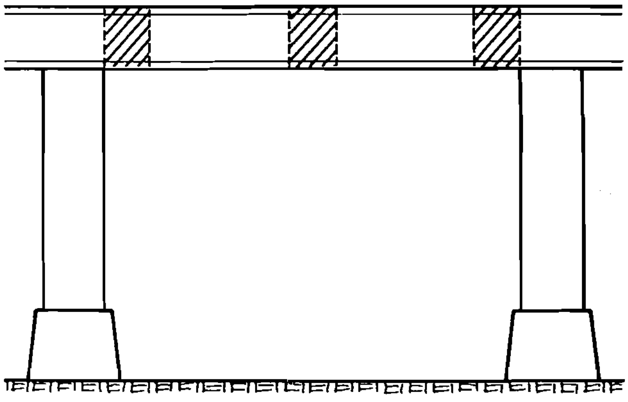


Fig. 31.17. Demolition of a bridge built with prefabricated beams or spans.

The minimum width of the zone to be fragmented should be of three rows of blastholes, and if the bridge is built with several arches it is recommended that the demolition be carried out simultaneously because whatever is left after each blast might be damaged or cracked by the same and constitute a high collapse risk.

Steel bridges on abutments and pillars of masonry

This type of bridge is very common in railways and implies dismantling the whole metal structure by conventional methods or by blasting with plaster charges as long as the throw does not constitute a risk; and blasting the pillars and abutments afterwards.

The latter structural elements can be drilled with vertical blastholes, if there is a usable access, or with horizontal holes as explained before.

Concrete bridges

Within this group, which is the most recently constructed, two types are seen: the arch beam and of prefabricated beams, or spans.

For the first, the demolition will be carried out by making cuts with confined explosive charges at strategic points of the structure, as indicated in Fig. 31.16.

As for span bridges, constructed with pretensed concrete beams, demolition is carried out in two different phases: first by blasting the beams, according to indications, in three zones of the span, Fig. 31.17, and followed by the pillars using steel cables, if necessary, to achieve their collapse. The blast should have at least two rows of holes and a powder factor of around 1.5 kg/m^3.

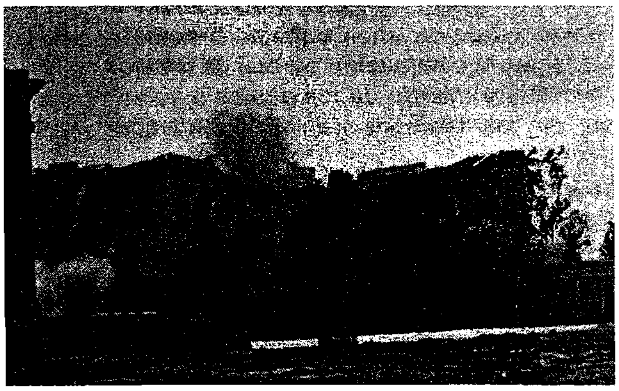


Photo 31.5. Demolition of a two-storey building of masonry.

31.5 DEMOLITION OF BUILDINGS

In the demolition of buildings with explosives, only basic principles can be given, as each case requires a complete project for the blast using general criteria and the application of specific criteria.

The types of structures that are usually demolished with explosives are basically constructed of brick, concrete or a mixture of both.

The work should commence with a detailed study of the building plans, if available, in order to determine:

- The existance of expansion joints.
- Areas that could affect the results of the demolition such as stairways, elevators, pipelines, basements, etc.

This phase should be complemented by a study of the construction materials, using the necessary scrapings for their identification, especially in the critical and doubtful areas. This work is equally or more important than the actual designing of the blasts, because there have been many cases in which, after only contemplating the building plans, the results have not been satisfactory because the real structures were not exactly the same. After the studies, the preparatory work must be carried out by:

- Separating the structural elements by mechanical means.
- Cutting the re-bars inside the reinforced concrete.
- Constraint of the elements with cables, bolts, etc.
- Elimination of the interior pannels, walls, door and window frames, etc.
- Elimination of metal beams and their substitution by props or supports, etc.

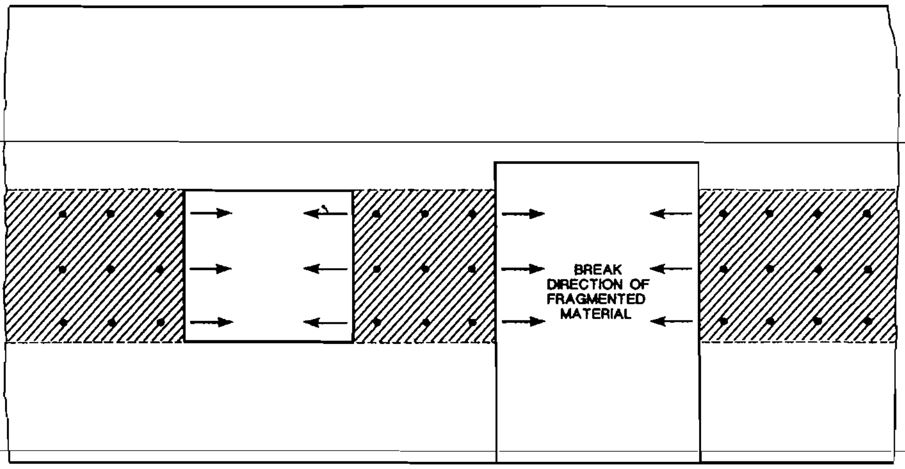


Fig. 31.18. Drilling in a brick wall.

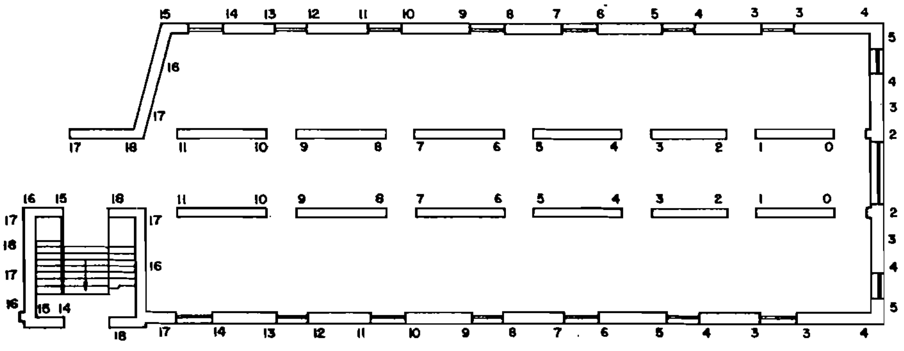


Fig. 31.19. Initiation sequence in a building of masonry.

The following gives an idea of the most frequent types of demolition.

31.5.1 *Buildings of masonry*

In these buildings, the first work is to eliminate the door and window frames and any other element which could interfere with demolition. Also, some of the interior pannels should be destroyed.

The blastholes are drilled in the sustaining walls following the patterns and recommendations of the preceding paragraphs.

The area to be blasted will be at the height of the windows, as mentioned before, so as to facilitate hand drilling of the holes, lower their number by using the spaces and improve the exit of the fragmented material, Fig. 31.18.

Although the objective of these blastings, owing to the characteristics of the construction materials, is the complete collapse of the building, the initiation sequences should be designed to achieve piling up of the debris at the same time so as not to affect the surroundings, streets, buildings, etc. and the maximum breakage of the structure during the fall.

In Fig. 31.19, a typical example of firing sequence is shown.

31.5.2 *Buildings of reinforced concrete*

As in the former case, the demolition should be preceded by preparatory work which consists in partially removing the interior partitions and the uncovering of the pillars that are to be blasted, opening grooves in the adjacent brick partitions, Fig. 31.20.

If the structure is complex and is next to other buildings that have to be respected, a careful study must be carried out of the static of the construction and the blast design.

The two most common types of demolitions are:
- One direction blasting.
- Demolition with collapse.

In the first instance, the elements to be blasted form a wedge, Fig. 31.21, with a face open towards the direction of the fall, and a sequence similar to that indicated.

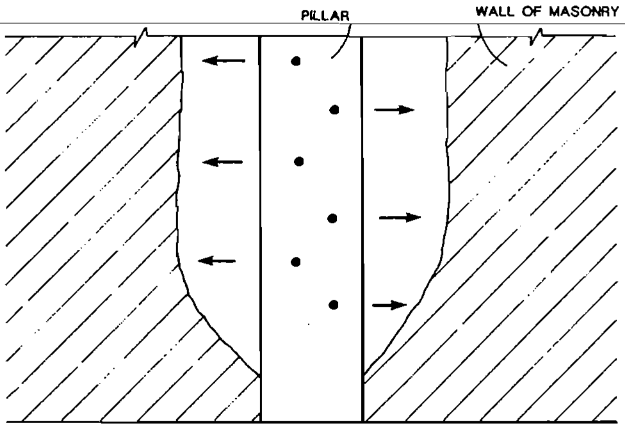


Fig. 31.20. Grooves opened in partitions that are joined to a pillar.

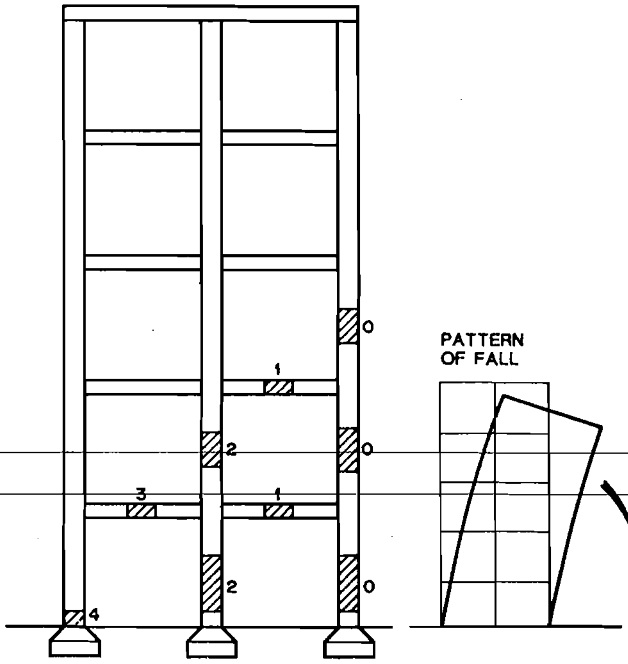


Fig. 31.21. Demolition of a building with one direction fall.

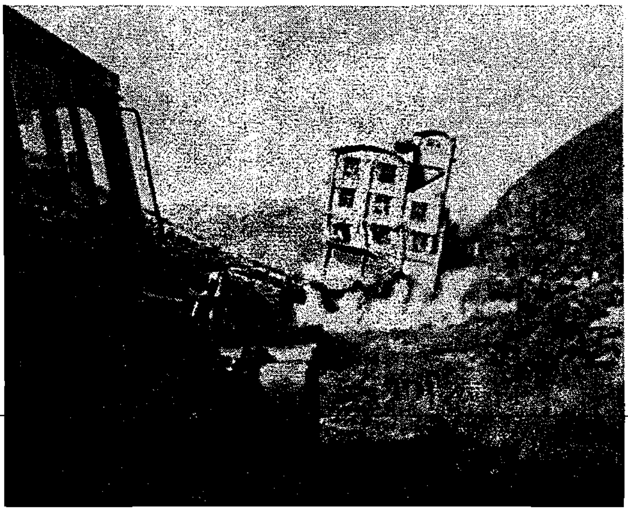


Photo 31.6. Demolition of a silo made of cement with fall in one direction.

initiation sequence is of vital importance, and for this reason the choice and use of the delay and milisecond delay detonators available should be based on an exhaustive analysis. The high-speed filming of the work is an important tool for the study and comprehension of the phenomena that developed and its results.

31.5.3 *Mixed buildings*

These buildings are made of pillars and beams of reinforced concrete and sustaining walls of masonry, both of which should be drilled according to the preceding paragraphs. Normally, this type of blast is with fall in one direction.

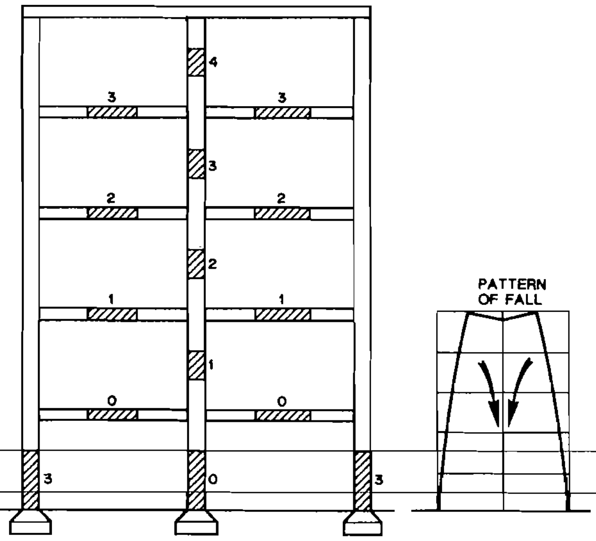


Fig. 31.22. Collapse demolition of a building.

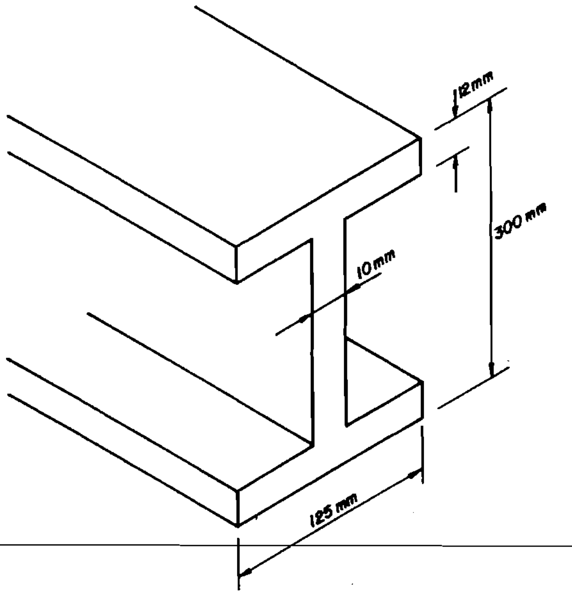


Fig. 31.23. Dimensions of the metal profile.

If the demolition is carried out with collapse upon the same occupied area, the initiation sequence and the blast pattern will be as indicated in Fig. 31.22.

If isolated structural elements exist which should exert some sort of force or pull during the blast, they should be held or anchored by different mechanical means, cables, bolts, etc.

On the other hand, when it is necessary to isolate some parts of the buildings, the uniting elements should be cut and, above all, the re-bars of the reinforced concrete.

It is easy to understand that in this type of blastings the

31.6 DEMOLITION OF STEEL STRUCTURES

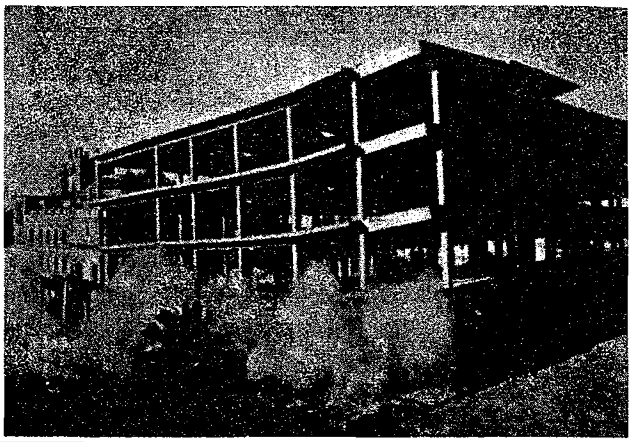


Photo 31.7. Demolition of the structure of a concrete building. (Courtesy of CAVOSA).

The demolition of steel structures with explosives is not as easy as those of concrete or masonry because the charges are not usually confined, and there is not such an intense phenomenon of self destruction during the fall of the buildings. Special precaution should be taken with the alterations produced by throw and airblast.

The equations that follow refer to a gelatin explosive in plaster charges, but without any special geometry.

a) *Sections of steel construction.* Double T beams, H beams, channels, compound steel spars, columns, plates:

$$Q = 34 \times A_{vg}$$

where: Q = Explosive charge (g), A_{vg} = Area of the transversal section (cm^2).

b) *Other steel sections.* Steels rich in carbon, chains, cables, reinforcing bars, high resistance elements, etc.

$$Q = 88 \times A_{vg}$$

where: Q = Explosive charge (g), A_{vg} = Area of transversal section (cm^2).

Example:

A double T steel beam with the dimensions that are indicated in Fig. 31.23, is to be cut.

Wing area = $2 \times 1.2 \times 12.5 = 30.0\ \text{cm}^2$
Area of the web = $(30 - 2.4) \times 1 = 27.6\ \text{cm}^2$
Total area $A_{vg} = 30.0 \times 27.6 = 57.6\ \text{cm}^2$
Explosive charge $Q = 34 \times 57.6 = 1858.4\ \text{g} \simeq 2\text{kg}$.

The plaster charges will be placed along the web of the beam and in contact with the wings. On occasions, plywood is placed on both sides to exert a shearing effect.

REFERENCES

Abad, M. et al.: *Voladuras Controladas en Zonas Urbanas.* IGME, 1985.

Berta, G.: *L'Explosivo Strumento Di Lavoro.* Italexplosive, 1985.

Gonzalez, E.: *Demolición de Chimeneas con Explosivos.* Rocas y Minerales.

Guftafsson, R. *Swedish Blasting Technique.* SPI, 1973.

ICI: *Blasting Practice.* 1972.

Lopez Jimeno, E.: *Voladuras de Chimeneas y Torres* (unpublished). 1978.

CHAPTER 32

Optimizing costs of fragmentation with drilling and blasting

32.1 INTRODUCTION

The main objective of fragmentation with explosives is to carry out the operation at the lowest possible cost, and at the same time comply with the technical specifications and safety conditions required.

In any mining project, drilling and blasting are the first basic operations that form part of an integrated system, Fig. 32.1, and can influence the results of the subsequent operations, in productivity as well as in costs.

Therefore, fragmentation and swell of rock should be studied with a global vision, relating it to the different parameters that correspond to the rest of the operations. In this chapter, the economical aspects of drilling and blasting in open pit production operations as well as the methods of predicting fragmentation and the basis of calculation for the construction of an economical model for optimization of the total costs of the operation will be analyzed.

32.2 ECONOMICAL ASPECTS OF DRILLING AND BLASTING

If the costs of drilling are analyzed and calculated by the method explained in the first chapters of this handbook, and they refer to the lineal meter of blasthole, it has been observed that for rotary percussive and rotary rigs the cost is proportional to the diameter of the drills, Fig. 32.2, although for the second system the gradient of increase is not as noticeable.

By taking into account the yield of breakage from the blast, also known as specific drilling, the drilling costs per cubic meter of rock vary according to the curves of Fig. 32.3. Four curves are given, which correspond to the respective combinations of rock drillability and blastability.

The interruption of the curves is due to the change from down the hole to rotary drilling.

As can be observed, the costs are reduced considerably as the diameter of the blastholes increases with the aforementioned change. It must not be forgotten that when drilling with tricone bits, the life of the bearings depends upon their size and so when the diameter is increased more thrust can be transmitted without affecting bit life, therefore increasing productivity and yield.

As to blasting costs, the following concepts are included:

- Explosives, for the bottom and column charges.
- Primers.
- Detonating cord.
- Electric detonators.
- Other accessories: blasting machines, connectors, lead wire and blasting ohmmeters, etc.
- Labor.

Apart from this, an additional cost must be taken into consideration due to secondary fragmentation of boulders or blocks of rock that cannot be handled by the loading, haulage and crushing equipment. An operation is considered acceptable when the volume of plaster shooting is below 5 % of the total volume broken in the primary blast.

The variation in blasting costs with increase in blasthole diameter differs from those of drilling, because part of the advantages are balanced out by some problems cited below:

The main advantages are:

- Increase in detonation velocity of the explosives which also increases Strain Wave Energy.
- Facility of using bulk explosives and mechanized charging, reducing costs in the price of explosives and in labor. This is the most important advantage.
- The planning and control of large blasts is easier and more complete than in smaller blasts, primarily due to the number of these required.

On the other hand, the problems are:

- The necessary powder factors are larger for the same size distribution because the charges are not as well distributed spacially inside the rock mass, and structural control of the rocks is increasingly important.
- The initiation and priming systems must be more trustworthy and efficient.
- Limitations because of loading and haulage capacity.
- Higher levels of vibrations and airblast.
- Higher risk of flyrock caused by the size of the fragments and the distance they cover.
- More intense fracturation of the remaining rock.
- Increase in ore dilution in mining operations.

Combining all these factors, the approximate costs of the blasting follow the tendencies reflected in Fig. 32.4.

From this it can be seen that the drilling diameter that gives minimum costs of fragmentation with explosives is not always the same, Fig. 32.5. The effect of scaled economy depends upon several parameters: rock properties, rate of production, organization of the work, environmental limitations, etc.

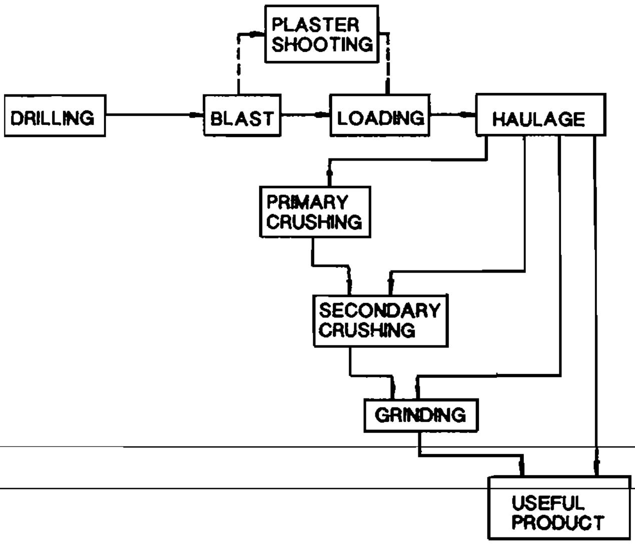


Fig. 32.1. Diagram of the mining cycle.

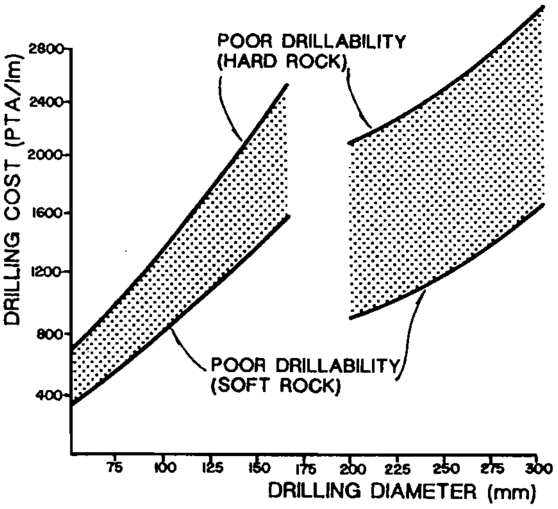


Fig. 32.2. Drilling costs per meter of blasthole.

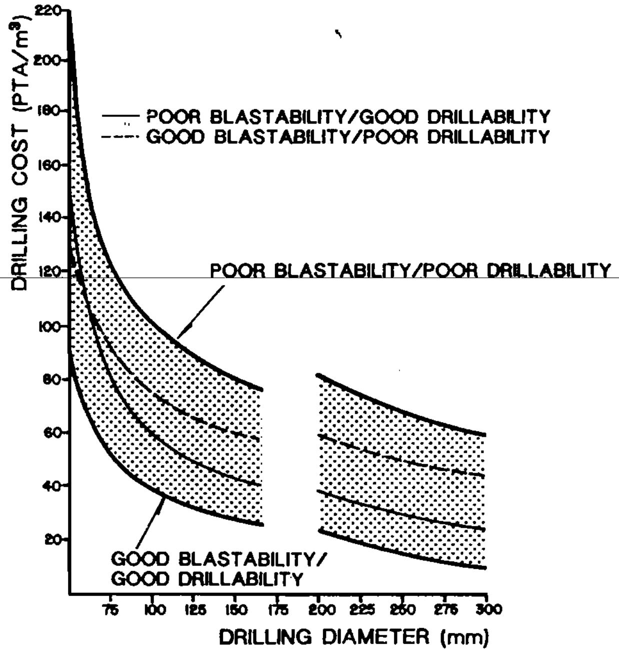


Fig. 32.3. Costs of drilling per cubic meter of blasted rock.

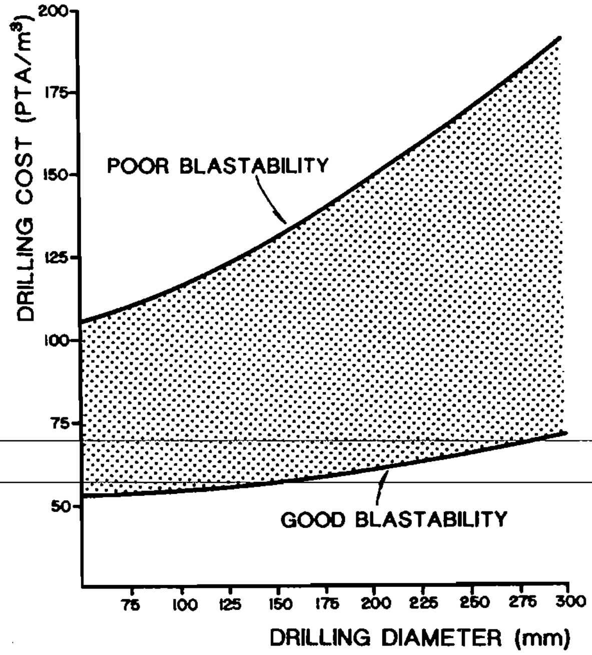


Fig. 32.4. Evolution of blasting costs without including drilling.

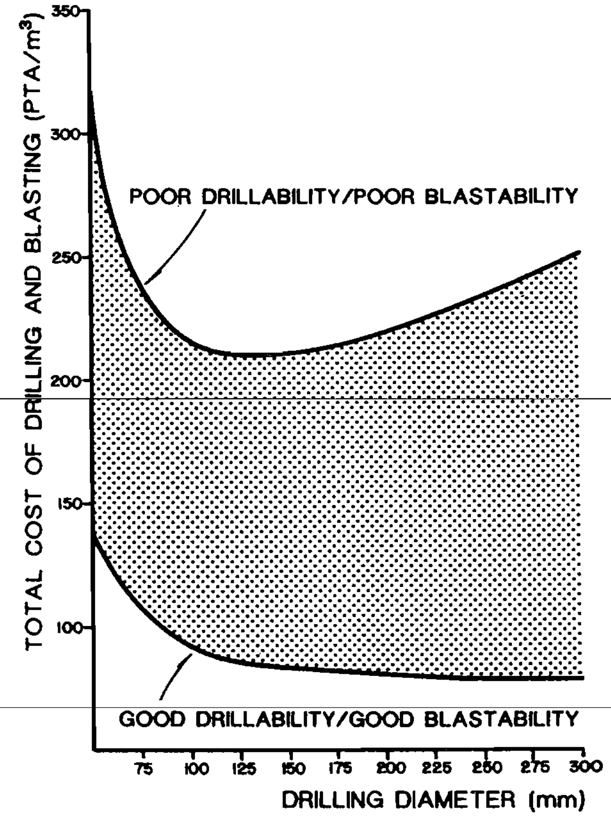


Fig. 32.5. Evolution of the total costs of drilling and blasting in function with the drilling diameter.

Photo 32.1. Panorama of an metal mine excavated by bench blasting.

Summing up, one can say that the large drilling diameters are interesting when the following conditions exist:

- High production rate.
- No limitations in capacity of loading, hauling and crushing equipment.
- No later comminution of the material is required, such as for wastes in mining.
- The rock mass is homogenous or intensely fissured, and there is no problem of toe or overbreak.
- No environmental problems, such as vibrations and airblast, etc.

32.3 MODEL FOR DETERMINING COST OPTIMIZATION

The optimization of fragmentation can be carried out by applying engineering systems to the creation of a global model of optimization. Roughly, the blocks of information given to the model are:

a) Characteristics of the rocks and rock mass
b) Properties of the explosives
c) Technical and economical information on drilling, loading, hauling and crushing equipment.

After calculating the drilling and charging patterns of the blastholes, a prediction of fragmentation and a simulation of the loading, hauling and crushing operations should be carried out, determining the productivity and unitary costs of each one.

Afterwards, a sensitivity analysis is carried out in order to calculate theoretically the optimum fragmentation, evaluating the system in practice by contrasting the actual size distribution and yields with those predicted in the calculations, Fig. 32.6.

Afterwards, the criteria for calculating cost extrapolation for the loading, hauling and crushing operations is given. For this, K_{80} is used, which corresponds with the grid opening through which 80% of the material passes.

32.3.1 *Loading costs*

The correlation between the degree of fragmentation and the productivity of the loading equipment is, according to Nielsen, the following:

$$\frac{K_{80}\,(\text{Standard})}{K_{80}\,(\text{Actual})} = \left[\frac{\text{Rend. } K_{80}\,(\text{Actual})}{\text{Rend. } K_{80}\,(\text{Standard})}\right]^{N}$$

If N is given a high value, the productivity of the loading equipment will not vary much with fragmentation. A high value of N can also be used to reflect an operation with a low coefficient of loading machine use due to a reduced number of dump trucks.

The loading costs are calculated with a rate equation:

$$C_c = \frac{\text{Fixed Costs}}{\text{Yield}} + \frac{\text{Yield}}{\text{Variable costs}} \times \left[\frac{\text{Standard Yield}}{\text{Yield}}\right]^{M}$$

According to Zeggeren and Chung, the loading costs respond to a function of the rate.

$$C_c = a_1 \times \left[\frac{K_{80}\,(\text{Actual})}{K_{80}\,(\text{Standard})}\right]^{0.5}$$

In practice, it has been demonstrated that fragmentation tends to be optimum when the relationship between the shovel capacity and the size of the block is between 6 and 7.

Another procedure used to estimate loading costs consists in calculating the productivity of the equipment from shovel size and the time invested in each loading and unloading cycle. The shovel fill factor is determined from the relative size RS defined by Adler (1986):

$$RS = AS/C$$

where: AS = Absolute block size, very small > 5 cm, medium 5-30 cm, very large 90-300 cm; C = Critical shovel size of the excavating equipment, in relation to any one of the sides of the ladle which is approximately in cubic form.

The fill factors that correspond with each relative size are indicated in Table 32.1.

32.3.2 *Hauling costs*

As the size of the blocks increases, the loading of the dump trucks becomes more difficult, not only affecting the quantity loaded but also the waiting, maneuvering and dumping times. The variation of the hauling cost with the fragmentation can be expressed by:

Table 32.1. Fragmented or fractured rock.

Description	Relative size	Fill factor
Very well blasted, floors	≤ ⅛	0.8-1.0
Well blasted	¼ – ⅛	0.7-0.9
Heavy blocks of pre-blasted rock	¼ – ½	0.5-0.8
Poorly blasted or fragmented rock	≥ ½	0.4-0.6

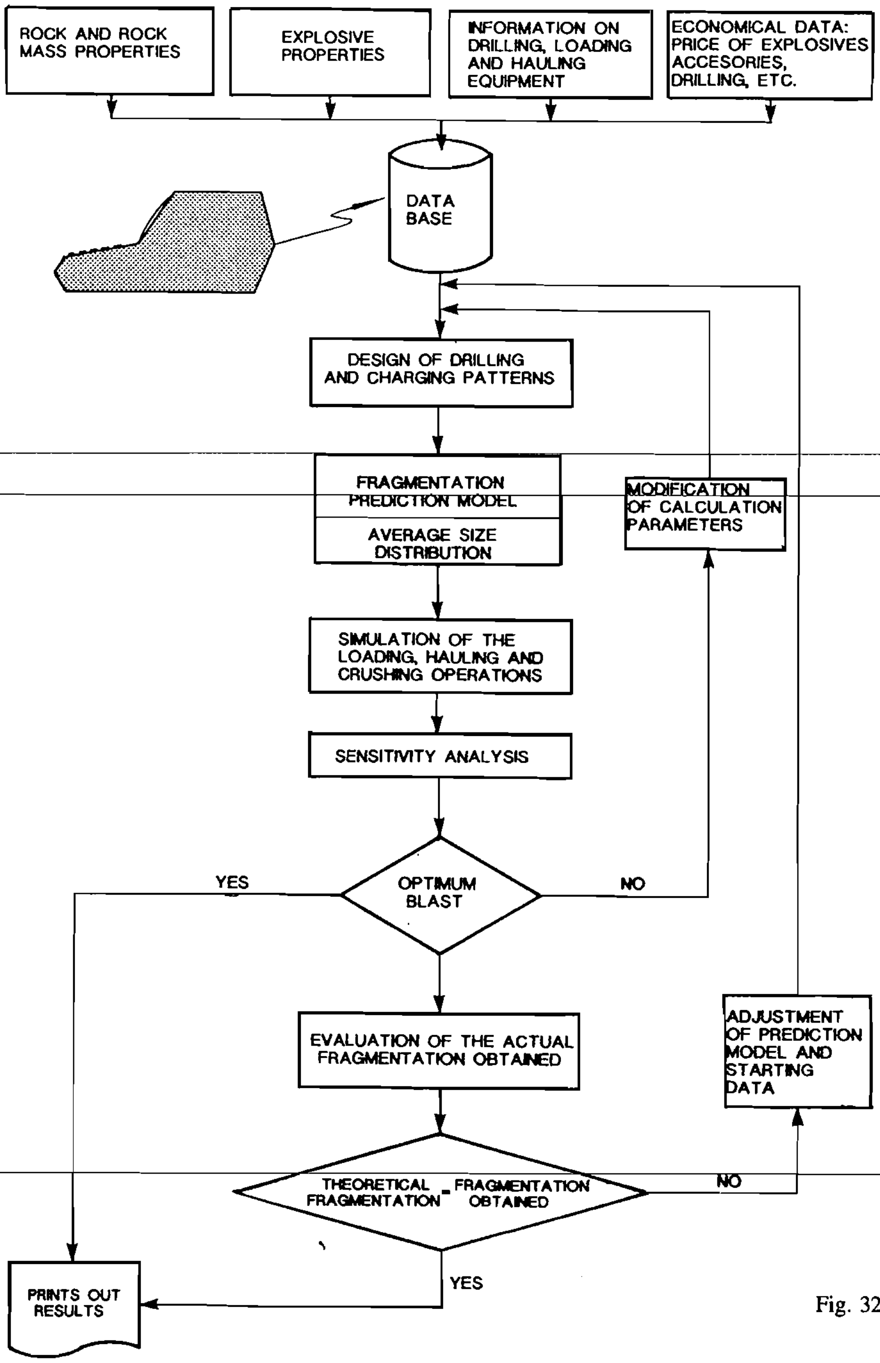

ROCK AND ROCK MASS PROPERTIES
EXPLOSIVE PROPERTIES
INFORMATION ON DRILLING, LOADING AND HAULING EQUIPMENT
ECONOMICAL DATA: PRICE OF EXPLOSIVES ACCESORIES, DRILLING, ETC.
DATA BASE
DESIGN OF DRILLING AND CHARGING PATTERNS
FRAGMENTATION PREDICTION MODEL
AVERAGE SIZE DISTRIBUTION
MODIFICATION OF CALCULATION PARAMETERS
SIMULATION OF THE LOADING, HAULING AND CRUSHING OPERATIONS
SENSITIVITY ANALYSIS
YES
OPTIMUM BLAST
NO
EVALUATION OF THE ACTUAL FRAGMENTATION OBTAINED
ADJUSTMENT OF PREDICTION MODEL AND STARTING DATA
THEORETICAL FRAGMENTATION = FRAGMENTATION OBTAINED
NO
YES
PRINTS OUT RESULTS

Fig. 32.6. Basic structure of the cost optimization model.

$$C_t = a_2 \times \left[\frac{K_{80}\,(\text{Actual})}{K_{80}\,(\text{Standard})}\right]^{0.5}$$

Crushing costs

As the cost of energy for this operation is very high, the variation of costs with fragmentation can be established from:

$$C_{tr} = a_3 + a_4 \times \left[\frac{K_{80}\,(\text{Actual})}{K_{80}\,(\text{Standard})}\right]^{0.5}$$

This relationship is based upon the supposition that the product of the primary crusher does not depend upon distribution size on entry.

32.4 PREDICTING THE FRAGMENTATION

The degree of fragmentation is a generic term used to describe the size distribution of the blasted material.

According to the type of project that is to be developed, the required size distribution of the rock can be very different. For example in open pit mining, although it is not desirable to break up the waste too much, the mineral should be quite fragmented if it is to be crushed and ground afterwards; in rip-rap quarries, the greatest percentage of the rock volume should correspond to large sizes, etc.

A number of controlable parameters influence fragmentation, as well as the rock properties themselves. The optimization of any operation of breakage by explosives requires a previous knowledge, even if it is approximate, of the blasted rock sizes under determined conditions.

There is no method or equation that gives an exact prediction, but during the past few years numerous investigators have developed models that go from simple empirical equations to complete computerized simulations.

In the following, some of the most widely accepted models today are exposed.

32.4.1 *Larsson's equation*

The equation proposed by Larsson (1973) for the determination of K_{50}, or the opening of the quadratic grid through which 50% of the the blasted material passes is:

$$K_{50} = s \times e^{(0.58 \times \ln B - 0.145 \times \ln (S/B) - 1.18 \times \ln (CE/c) - 0.82)}$$

where: B = Burden (m); S/B = Ratio Spacing/Burden; CE = Specific charge (kg/m^3); c = Rock constant. It is the equivalent of the powder factor of gellatin dynamite necessary to fragment the rock, normally between 0.3 and 0.5 kg/m^3; s = Blastability constant. This is a factor that takes into account the heterogeneity and the discontinuities of the rock mass (Very jointed and fissured rock s = 0.60, Jointed rock s = 0.55, Normal rock with hair cracks s = 0.50, Relatively homogeneous rock s = 0.45, Homogeneous rock s = 0.40).

With the nomographs of Figs 32.7, 32.8, and 32.9, apart from K_{50}, the curves of the theoretical size distribution of the blasted material can be estimated, from a given blast design or by reversing the procedure, determining the design from the desired fragmentation.

32.4.2 *The formula from SVEDEFO (Swedish Detonic Research Foundation)*

The original formula proposed by Larsson does not take into account the effect of bench height and stemming height, but SVEDEFO has added a term that makes the final equation as follows:

$$K_{50} = s \times \left[1 + 4.67 \left(\frac{T}{L}\right)^{2.5}\right] \times e^{0.29 \ln B^2 \sqrt{\frac{S}{1.25}} - 1.18 \ln \left[\frac{CE}{c}\right] - 0.82}$$

The new parameters are: T = Stemming length (m), L = Depth of Blasthole (m).

32.4.3 *KUZ-RAM Model*

This model was developed by Claude Cunningham (1983), of AECI, South Africa, based upon the size distribution curve of Rosin-Rammler and the empirical equation of the average fragment size obtained from the blast given by V.M. Kuznetsov.

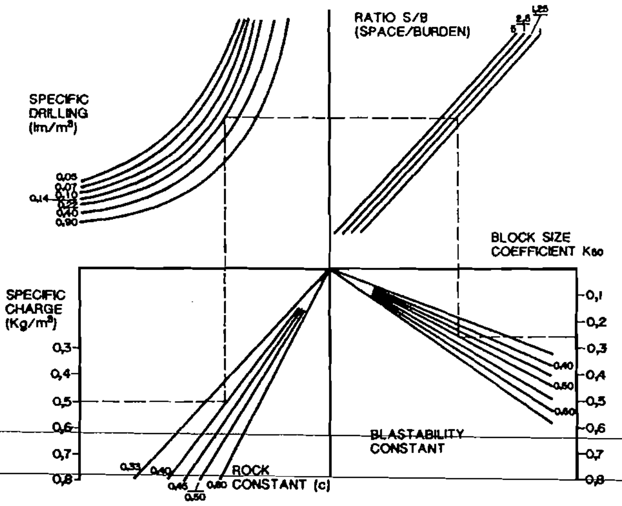


Fig. 32.7. Nomograph for determining block size.

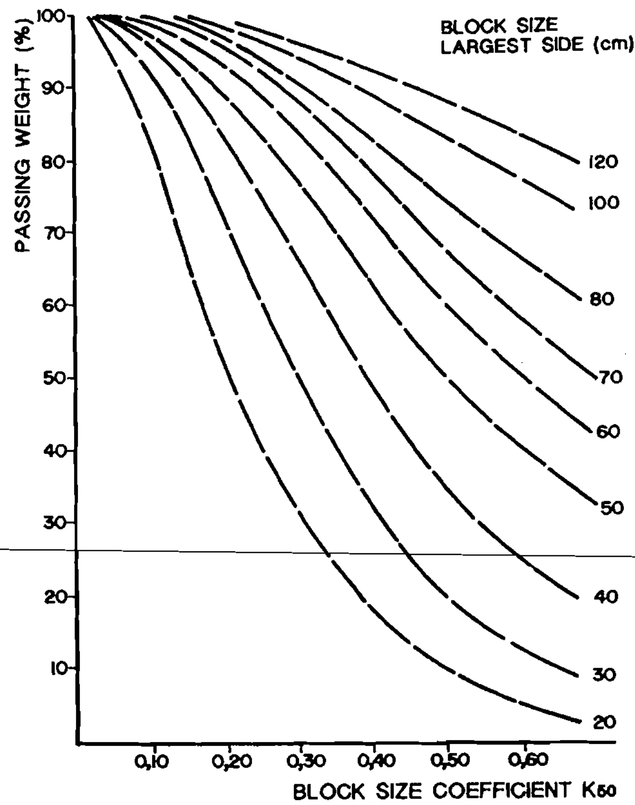


Fig. 32.8. Curves for the theoretical block size of blasted rock for various K_{50} values.

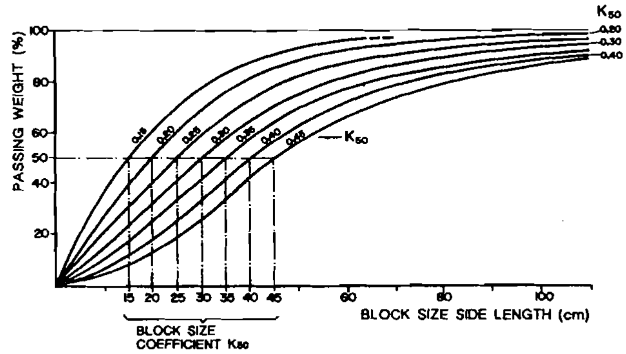


Fig. 32.9. Determination of the coefficient of the Block Size K_{50} of the fragmented material.

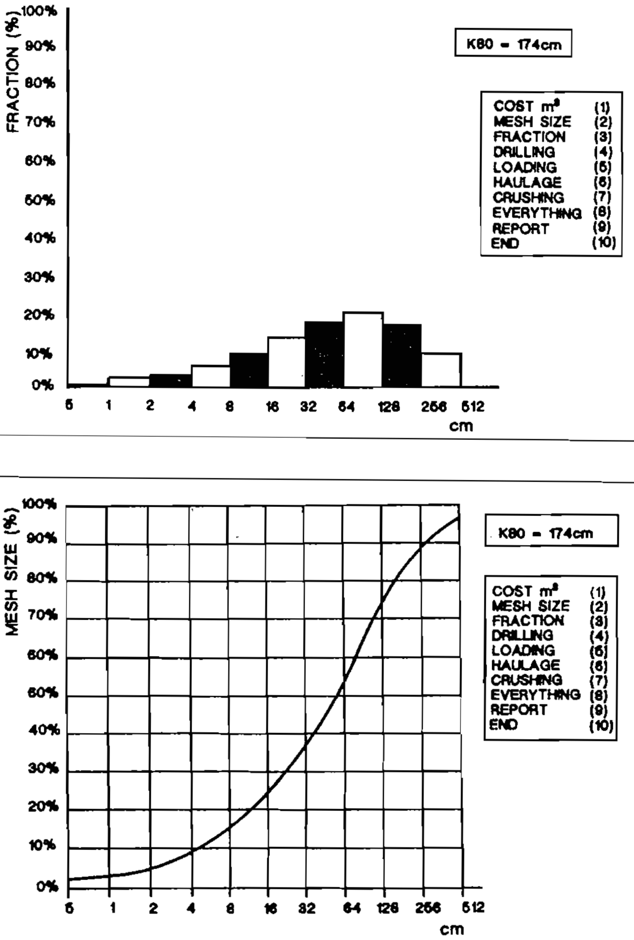


Fig. 32.10. Prediction of size distribution and calculation of the K_{80} of the blasted rock with the KUZ-RAM program.

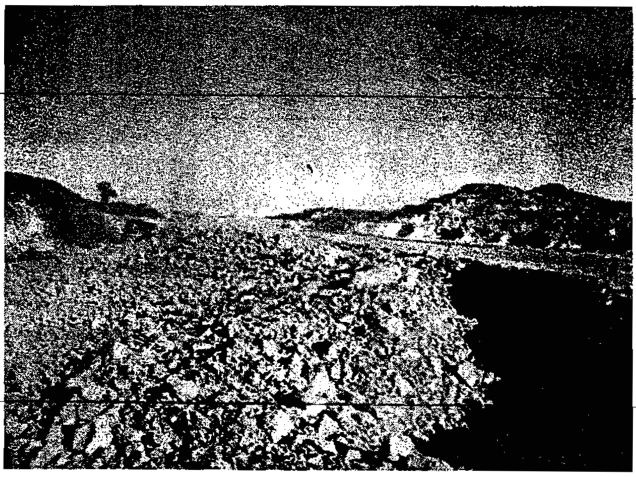


Photo 32.2. General aspect of a large size bench blast.

Table 32.2.

Parameter	*u* increases as parameter
B/D	Decreases
L/H	Increases
S/B	Increases
Staggered pattern	Increases
Drilling precision	Increases

The development of the model is as follows:

a) *Kuznetsov equation*

$$\overline{T}_b = F_r \times \left[\frac{VR_o}{Q}\right]^{0.8} \times Q^{1/6}$$

where: $\overline{T}_b$ = Average fragment size (cm), F_r = Rock factor (Very soft rocks $F_r = 3$ ($f = 3$ to 5); Soft rocks $F_r = 5$ ($f = 5$ to 8); Medium rocks $F_r = 7$ ($f = 8$ to 10); Hard fissured rocks $F_r = 10$ (10 to 14); Hard homogeneous rocks $F_r = 13$ (12 to 16); (f = Protodyakonov Factor)), VR_o = Rock volume broken per blasthole, taken as burden × spacing × height (m³), Q = Mass of TNT which is equivalent in energy to that of the explosive charge in each blasthole (kg), Q_b = Explosive charge in the blasthole (kg), PRP = Relative Weight Strength of this explosive, (ANFO = 100, TNT = 115).

$$Q_b \times \text{PRP} = Q \times 115$$

$$Q = \frac{Q_b \times \text{PRP}}{115}$$

The equation of Kuznetsov is converted into:

$$\overline{T}_b = F_r \times \left[\frac{VR_o}{Q_b}\right]^{0.8} \times Q_b^{1/6} \times \left[\frac{\text{PRP}}{115}\right]^{-19/30}$$

$$\overline{T}_b = F_r \times CE^{-0.8} \times Q_b^{1/6} \times \left[\frac{115}{\text{PRP}}\right]^{19/30}$$

where: CE = Powder factor (kg/m³).

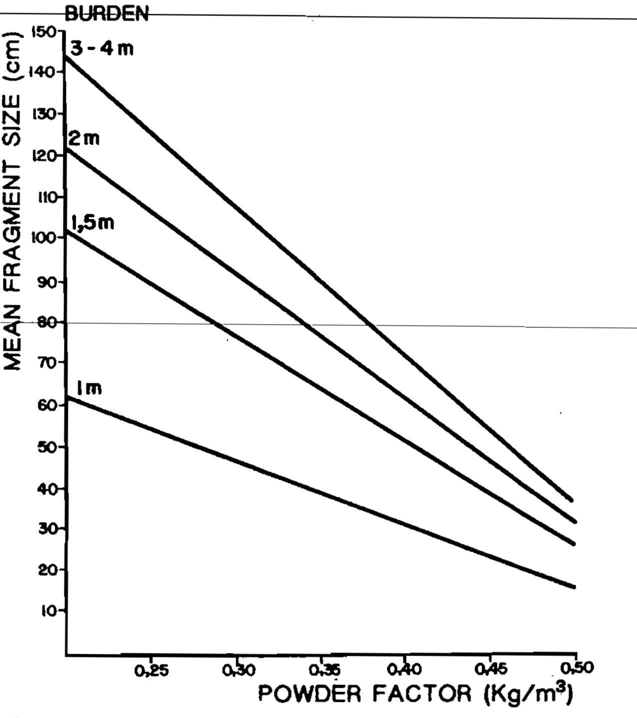


Fig. 32.11. Average fragment size in function with burden and powder factor.

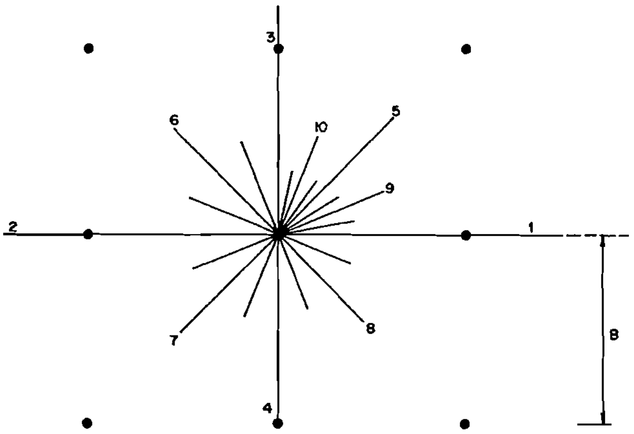


Fig. 32.12. Crack pattern produced by an explosive charge in a homogeneous rock with a square pattern $B = S$.

b) *Equation of Rosin-Rammler*

$$PC = e^{-(T_b/T_{bc})^u}$$

where: PC = Proportion of the material retained on screen T_b, T_b = Screen size, T_{bc} = Characteristic size, u = Index of uniformity.

As the equation of Kuznetsov gives the mean value of T_b, then $PC = 0.5$, so that:

$$0.5 = e^{-(T_b/T_{bc})^u}$$

from which:

$$T_{bc} = \frac{T_b}{(0.693)^{1/u}}$$

c) *Influence of the blast design upon u*

The value of u determines the size distribution curve and usually varies between 0.8 and 2.2. A high value indicates a uniform fragmentation while the smaller values reflect a large quantity of fines as well as oversize.

The influence of the different design parameters upon u is shown in Table 32.2.

The algorithm to calculate u, according to Cunningham (1987) is as follows:

$$u = [2.2 - 14B/D] \times [((1 + S/B)/2)^{0.5}] \times$$
$$[1 - E_p/B] \times [\,|l_f - l_c|\,/l + 0.1]^{0.1} \times \frac{l}{H}$$

where: D = Blasthole diameter (mm), B = Burden (m), S = Spacing (m), l = Total charge length (m), l_f = Length of bottom charge (m), l_c = Length of column charge (m), H = Bench height, E_p = Typical deviation due to drilling error.

If the blasts are designed with a staggered pattern, the calculated value for u should be increased by 10%.

d) *Limitations of the Kuz-Ram Model*

Certain precautions should be taken with this model such as:

– The S/B relationship applies to the drilling pattern and not to the initiation sequence. It should be no larger than 2.

– The initiation sequence and the delay timings should be such as to give good fragmentation without producing misfires or cut-offs.

– The explosive should yield energy close to its calculated Relative Weight Strength.

– The jointing and homogeneity of the mass requires careful study, especially when the spacing between jointing is smaller than the distance between holes.

32.4.4 The Dinis da Gama formula (1970)

This equation considers blasting as a comminution process in which the fragmented material satisfies the following size distribution law:

$$PC = a \times W^b \times \left[\frac{T_b}{B}\right]^c$$

where: PC = Percentage of accumulated material that is smaller than the fraction of size T_b, W = Necessary energy for fragmentation (kWh/t),

$$W = 10 \times \frac{W_i}{\sqrt{K_{80}}}$$

W_i = Bond's rock index, K_{80} = The size which 80% of the material passes, B = Size of the burden in the blasting pattern, a, b and c = Constants which depend upon the properties of the rocks and the explosive.

The values of the constants are obtained by resolving a system of three equations with three unknown quantities established from the data of the size distribution curve of a representative sample of blasted material.

32.4.5 *Nomograph of Gustafsson*

In this subject, Gustafsson contributes a nomograph that is valid for small diameter drilling, with which the mean size of the fragments can be estimated from the specific charge and the size of the burden.

32.4.6 *Bi-dimensional computerized model*

For quite a few years, the ICI group has been using a computer program to predict fragmentation that is included within the SABREX system (Scientific Approach to Breaking Rock with Explosives).

The method is based upon creating a model in which the geometry of the blastholes and the radial cracks formed around each one are represented upon a horizontal plane which perpendicularly intersects the explosive columns.

The algorithm used to calculate the number of radial cracks around each charge and at a determined distance from these is that of Harries (1973):

$$N_o = \frac{\sigma_b}{RT'}$$

where: N_o = Number of cracks around the charge,

σ_b = Strain in the blasthole wall, RT' = Dynamic tensile strength of the rock.

At a distance DS from the axis of the blasthole, the number of cracks will be N:

$$N = \frac{N_o}{DS/b} \times e^{-\alpha \frac{DS}{b}}$$

where: b = Radius of the blasthole, α = Coefficient of absorption which usually varies between 0.002 and 0.008.

The geomechanic parameters of the rock which are used in the modelization are:

- Density
- Young's Modulus

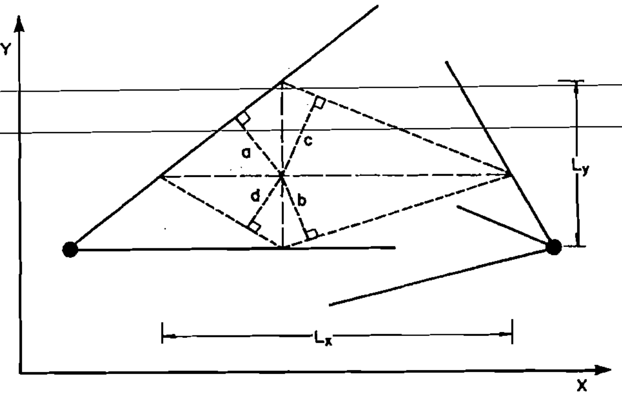


Fig. 32.13. Determination of the block size between radial cracks.

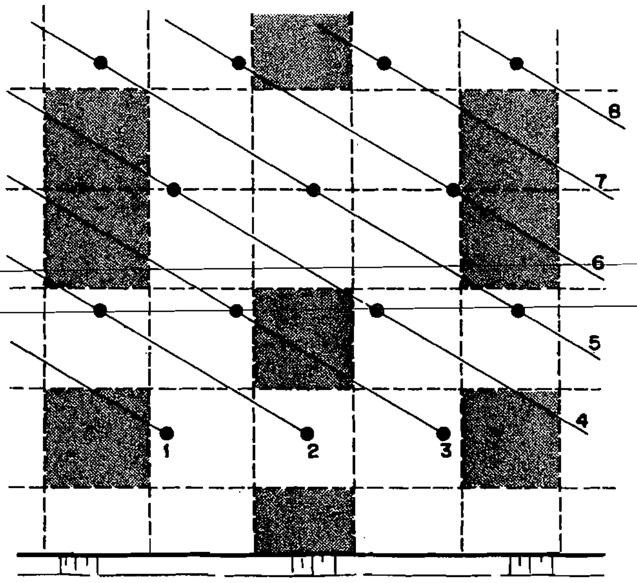


Fig. 34.14. Modelization of the discontinuities in the rock mass.

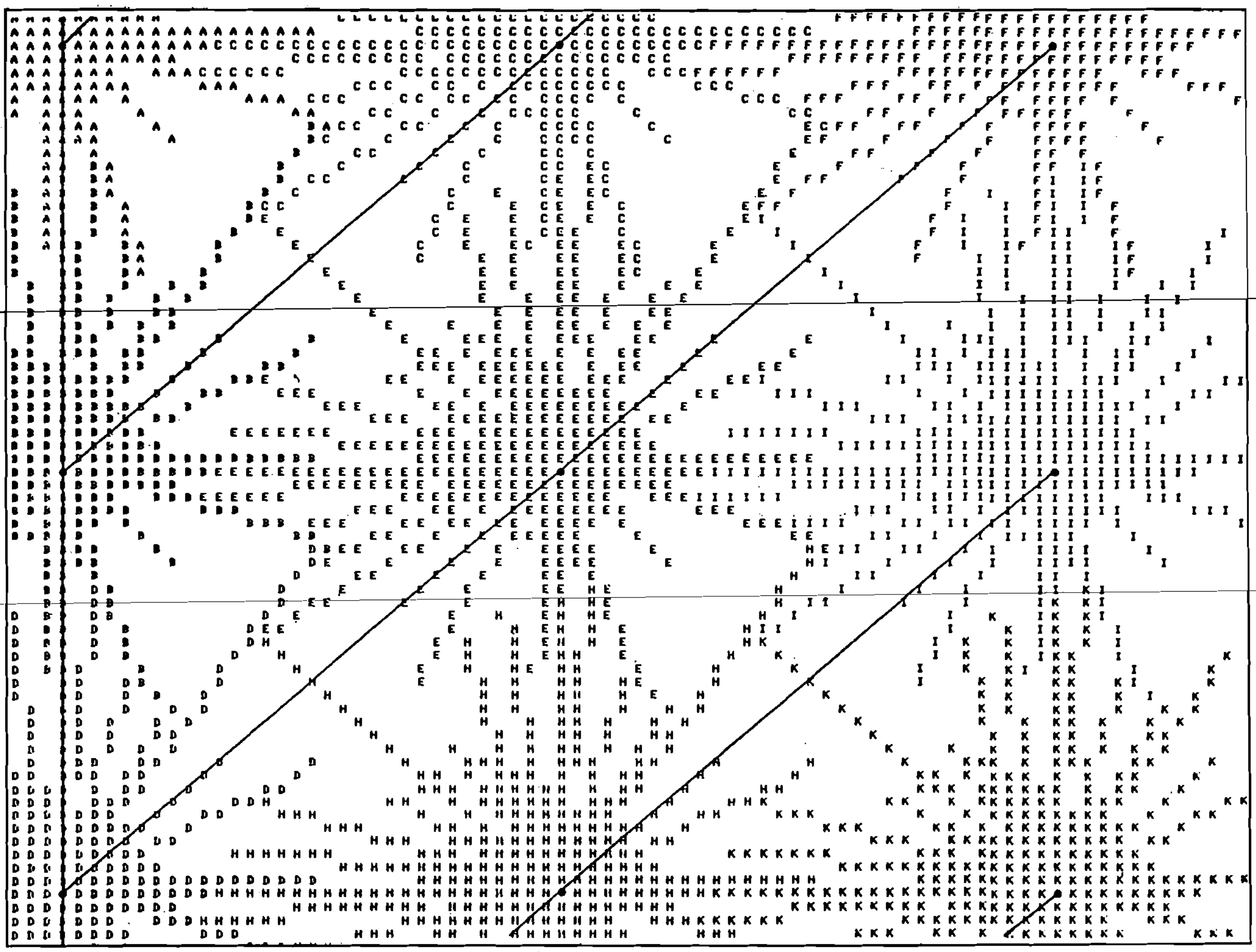


Fig. 32.15. Graphic representation of the cracks originated around each blasthole for a determined pattern and initiation sequence.

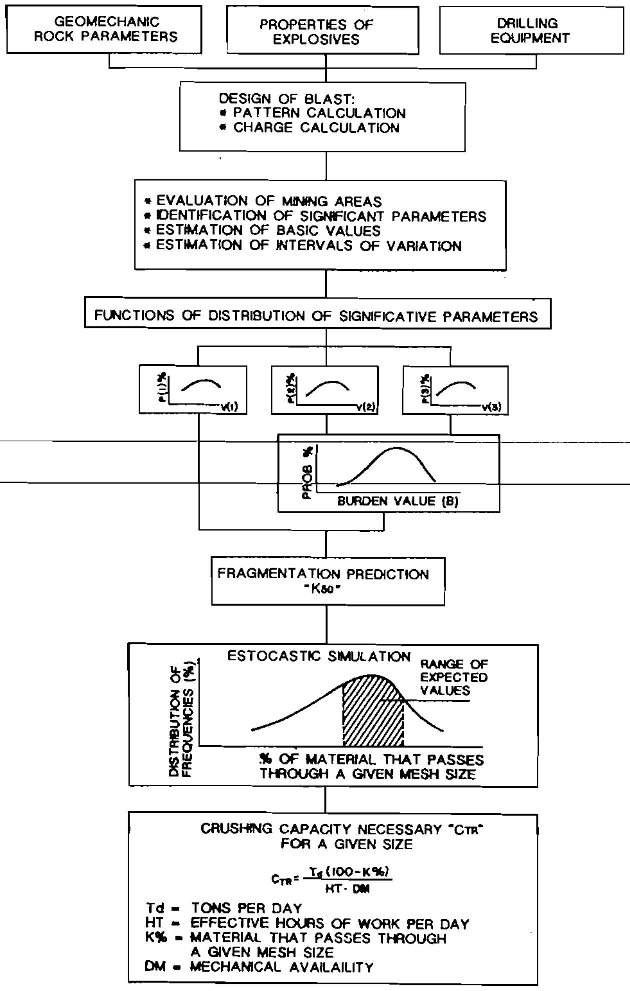

GEOMECHANIC ROCK PARAMETERS
PROPERTIES OF EXPLOSIVES
DRILLING EQUIPMENT
DESIGN OF BLAST:
• PATTERN CALCULATION
• CHARGE CALCULATION
• EVALUATION OF MINING AREAS
• IDENTIFICATION OF SIGNIFICANT PARAMETERS
• ESTIMATION OF BASIC VALUES
• ESTIMATION OF INTERVALS OF VARIATION
FUNCTIONS OF DISTRIBUTION OF SIGNIFICATIVE PARAMETERS
P(1)%
V(1)
P(2)%
V(2)
P(3)%
V(3)
PROB %
BURDEN VALUE (B)
FRAGMENTATION PREDICTION "K50"
ESTOCASTIC SIMULATION
RANGE OF EXPECTED VALUES
DISTRIBUTION OF FREQUENCIES (%)
% OF MATERIAL THAT PASSES THROUGH A GIVEN MESH SIZE
CRUSHING CAPACITY NECESSARY "CTR" FOR A GIVEN SIZE
CTR = Td (100-K%) / HT · DM
Td = TONS PER DAY
HT = EFFECTIVE HOURS OF WORK PER DAY
K% = MATERIAL THAT PASSES THROUGH A GIVEN MESH SIZE
DM = MECHANICAL AVAILAILITY

Fig. 32.16. Probabilistic fragmentation model for the selection of the primary crusher size (Borquez, 1981).

- Poisson's Ratio
- Compressive strength
- Tensile strength
- Porosity.

The basic data of the explosive is the strain induced on the blasthole wall that is a function of the density of the explosive, the detonation velocity and the coupling of the charge

The orientation of the cracks is carried out in a random manner from each hole and their propagation by the opening action of the gases is thought to stop when they reach a free face or a maximum length equal to the size of the burden.

In multiple sequenced blastings, the propagation of cracks in adjacent holes is paralyzed at the moment the cracks intersect. This way, the effect that the same blast pattern has with different initiation sequences can be studied.

After having generated the pattern of cracks from each blasthole, the evaluation of the fragmentation is carried out by the Monte Carlo method which consists in taking at random different points of the plane. From each one of these points the size of the block, shaped by radial cracks, included in that point is determined, as shown in Fig. 32.13.

By repeating this process a determined number of times, the size distribution curve can be determined.

Some interesting applications of this fragmentation simulation model are based upon the following studies:

- Effects of different explosives for the same drilling pattern.
- Influence of the initiation sequence upon fragmentation.
- Incidence of collaring errors in blastholes.
- Repercussion of misfires.
- Influence of pattern variation with the same charge.
- Effect of the rock mass structure upon the results of the blast.

In the last application, the process starts with a modelization of the discontinuities of the rock mass, as shown in Fig. 32.14.

If the drilling pattern is large with respect to the spacing of the natural joints, the size distribution of the muck will be affected by large blocks previously formed by said discontinuities. The influence of these fissured beds of the rock can be evaluated by considering the different types of fillings of the joints and consequently the different coefficients of absorption of the strain waves when they cross these planes.

In Fig. 32.15, an example of modelization for a determined pattern and initiation sequence can be seen.

32.5 PROBABILISTIC ANALYSIS OPTIMIZATION MODEL

The model mentioned before is based upon the supposition that all parameters intervening in the calculation of the burden and prediction of the fragmentation are considered known or with the most probable values.

However, those parameters are usually subject to some doubt and it is necessary to be able to evaluate their effect on the indicated calculations and estimations. This is by means of a probabilistic (risk) analysis method applied to test the fragmentation model.

The risk analysis combines the variabilities of all significant parameters that enter into consideration in order to obtain the probability of the estimated values and the relative risk that the predicted value will not be reached.

On the whole, these models begin with the identification of the most significant parameters, the assigning of a distribution of probabilities to each one, and the simulation by computer of a large number of cases, selecting a value randomly from the probability distribution in order to obtain the function of probabilities of fragmentation or any other design parameter. The most common procedure for simulation is the Monte Carlo technique, Fig. 32.16.

REFERENCES

Borquez, G.V.: *Estimated drilling and blasting cost – An analysis and prediction model.* E/MJ, January, 1981.

Borquez, G.V.: *Use of a risk analysis model to evaluate rock fragmentation by blasting and to determine the size of a primary crusher.* SME, September, 1986.

Cunningham, C.: *The Kuz-Ram Model for production of fragmentation from blasting.* First Symp. on Rock Fragmentation by Blasting, Lülea, 1983.

Da Gama, C.D.: *The size of the largest fragment in rock blasting.* Proceedings of the 3rd Congress Int. Soc. Rock Mechanics, Denver, 1974.

Gustafsson, R.: *Swedish Blasting Technique.* SPI, Gotherburg, 1973.

Harries, G. & J.K. Mercer: *The science of blasting and its optimization to minimize costs.* The AUS, IMM, June, 1975.

Hjelmberg, H.: *Some ideas on how to improve calculations of the fragment size distribution in bench blasting.* Nitro Nobel AB, 1983.

Kuznetzov, V.M.: *The mean diameter of the fragments formed by blasting of rock.* Soviet Mining Science, 1973.

Lopez-Jimeno, C. & L. Fernandez: *Optimización de los Costes Mineros a través del Control de las Voladuras. Programa OPTIVOL.* Canteras y Explotaciones. Diciembre, 1989.

Lopez Jimeno, E.: *Implantación de un Método de Cálculo y Diseño de Voladuras en Banco.* ETS de Ingenieros de Minas de Madrid. Tesis Doctoral, 1986.

Lownds, C.M.: *Computer modelling of fragmentation from an array of shotholes.* First Symp. on Rock Fragmentation by Blasting, Lülea, 1983.

Lundborg, N.: *Computer program for charge calculations by bench blasting.* Swedish Detonic Research Foundation.

Lundquist, R.G. & C.J. Konya: *A comparison of efficiency of mechanical and explosive comminution.* 24th US Symp. on Rock Mechanics, June 1983.

Nielsen, K.: *Sensitivity analysis for optimum open pit blasting.* SEE, 1985.

Van Zeggeren, F. & S.M. Chung: *A model for prediction of fragmentation, patterns and costs in rock blasting.* CIL, 1975.

CHAPTER 33

Land vibrations, air blast and their control

33.1 INTRODUCTION

The principal disturbances created by blasting are: vibrations, air blast and fly rock, Fig. 33.1. All of them can, under some circumstances, cause damage to structures nearby and, apart from this, be the source of permanent conflict with the inhabitants who live close to the operation. Dust formation is also quite frequent and difficult to control.

In order to solve these problems, it is necessary to have more highly qualified blast superintendents so that they can reduce the level of disturbances at a reasonable cost. Another issue to take into account is the job of information and public relations, which is becomming a necessity, undertaken by the directors of the operation. This can be even more effective than an exhaustive study by specialists in the matter.

This chapter analyzes the theory of vibration and air blast produced by blasting, the methodology of study, the applicable damage criteria and the design parameters which must be taken into consideration by the technician to be able to control these environmental alterations.

33.2 PARAMETERS WHICH AFFECT VIBRATION CHARACTERISTICS

The parameters which affect the characteristics of vibrations are, practically, the same ones which influence the results of the blasts. They can be classified in two groups: controllable and uncontrollable by the users of explosives.

The local surrounding geology and rock geomechanics have great influence on vibrations.

In homogeneous and massive rock masses the vibrations are propagated in all directions; but in complex geological structures, the wave propagation can vary with the direction and, consequently, give different attenuation indexes or laws of propagation.

When the rocky substratum is covered by soil overburden this usually affects the intensity and frequency of vibrations. Soil usually has less elasticity modulus than the rocks and, for this reason, the wave propogation velocity diminishes in this type of material. The vibration frequency *f* also dimishes, but displacement *A* increases significantly as the overburden thickens.

The magnitude of the vibrations decreases rapidly with distance increase if soil overburden is present because a large part of the energy is used up in overcoming friction between particles and in displacing them.

At points close to the blasts, the characteristics of the vibrations are affected by the factors of blast design and their geometry. At large distances from the blast, the design factors are less critical and the transmitting medium of rock and soil overburden dominates the wave characteristics.

The surface materials modify the wave trains making these last longer and have lower frequencies, therefore increasing the response and potential damage to nearby structures.

From a study carried out by Stagg and Dowding (1980), it can be deduced that the vibration frequencies in coal mines are lower than those generated in quarries and construction jobs, Fig. 33.2, which is justified by the long length of the explosive columns, the complexity of the geological structures and by the presence of soil overburden.

An appreciable amount of the energy transported by vibrations in coal mines has a frequency that is lower than 10 Hz. This induces important ground displacement and high stress levels, which provoke damages in structures with resonance frequency between 4 and 12 Hz.

In another statistical study on more than 2700 registers carried out by Nobel's Explosive Company Limited, it can also be observed that 90% of the blasts in coal mines produces vibration frequencies under 20 Hz. The number of blasts in quarries that give frequencies between 4 and 21 Hz is approximately 80%, Fig. 33.3.

The phenomenon of low frequencies is most clearly seen in underwater blasts or in rock masses that are saturated with water.

33.2.2 *Charge weight per delay*

The magnitude of ground and air vibrations at a determined point varies with the explosive charge that is detonated and the distance of that point from the blast area. In blasts where more than one period number of detonator is used, the largest charge per delay has the most direct influence on vibration intensity and not the total charge used for the blast, as long as the delay interval is sufficient to avoid constructive interferences between the waves generated by the different groups of blastholes.

When there are various blastholes in a blast with detonators which have the same nominal delay time, the

Photo 33.1. Alterations produced by blasts: vibrations, air blast, fly rock and dust.

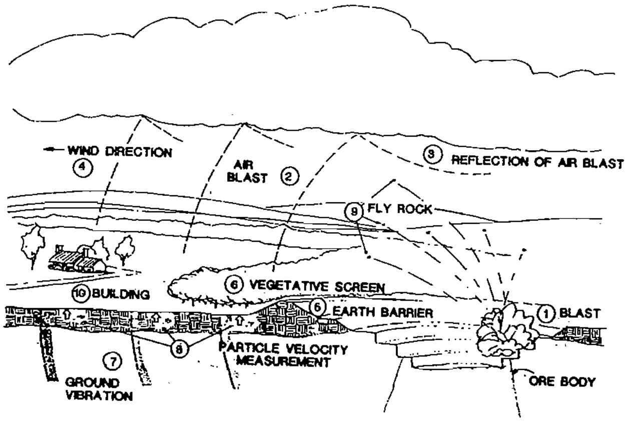


Fig. 33.1. Disturbances originated by rock blastings.

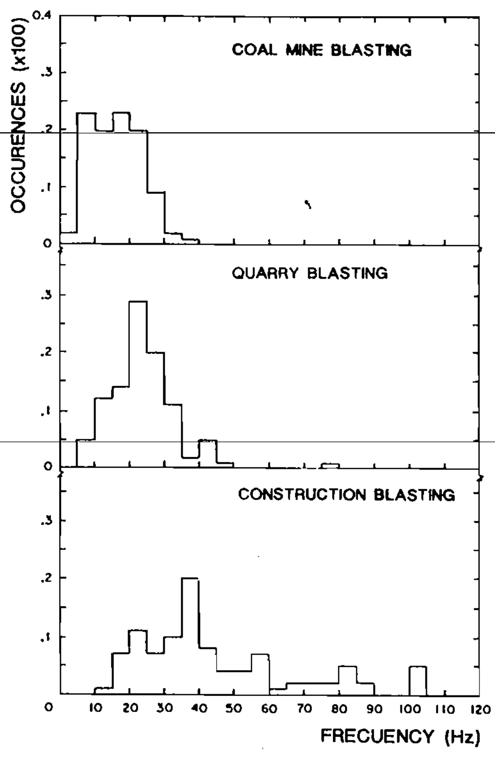


Fig. 33.2. Predominating frequencies of vibrations from coal mine, quarry, and construction blasting (Dowding et al. 1980).

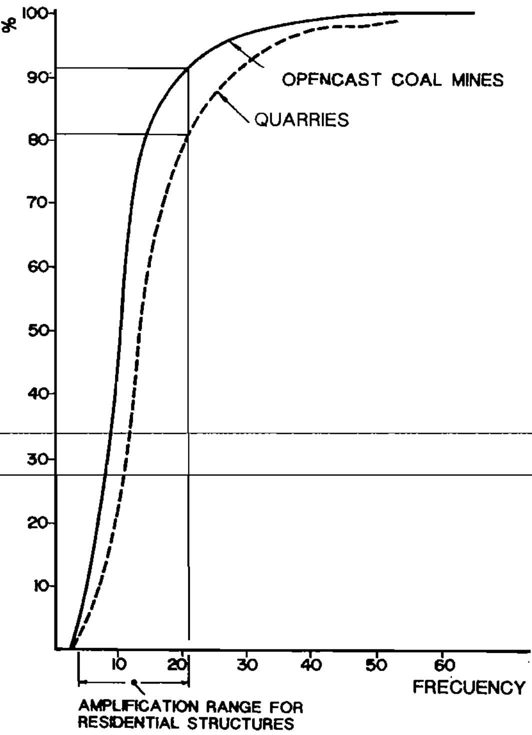


Fig. 33.3. Predominant vibration frequencies in surface coal mines and quarries.

Table 33.1. Cooperation fractions for different types of detonators.

Detonator	Period number	Period (ms)	Scatter (ms)	Cooperation within period (Reduction factor)
VA-MS/Nonel	1-10	25	5-10	½
VA-MS/Nonel	11-20			⅓
VA-MS/Nonel	24-80	100	20- 50	¼
VA/MS	1-12	500	100-200	⅙

Note: These values are only for frequencies over 20 Hz.

maximum charge weight per delay is usually less than the total, owing to cap scatter in the break times of each detonator system. For this reason, in order to determine said charge weight per delay, a fraction of the total number of charges initiated by detonators of the same nominal delay is estimated. Thus, for example, for the detonators manufactured by Nitro Nobel AB the following cooperation fractions can be estimated (Persson, 1980) Table 33.1.

The charge weight per delay is the most important individual factor that affects the generating of vibrations. The relationship that exists between vibration intensity and the charge is of potential type, therefore, for particle velocity the following exists:

$$v \propto Q^a$$

The investigations carried out by the US Bureau of Mines show that the value of *a* is around 0.8.

33.2.3 *Distance from point of blast*

The distance from the blasts has, as happens with the charge, great influence on the magnitude of vibrations.

As the distance increases, vibrations diminish according to a law of the following type:

$$v \propto \frac{1}{D^b}$$

where the value of b, according to the US Bureau of Mines, is around 1.6.

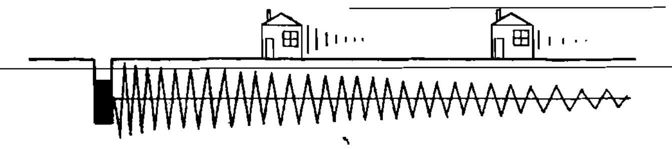
Another effect of distance is due to attenuation of the high frequency wave components, as the earth acts a filter through which the lower frequencies pass. Thus, at long distances the ground vibrations will have more energy in the low frequency range, Fig. 33.4.

33.2.4 *Powder factor*

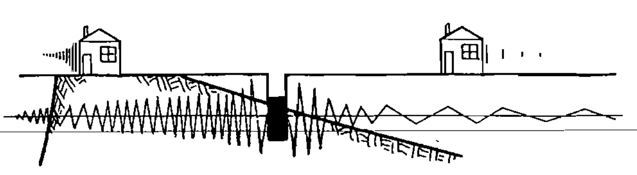
Another interesting and sometimes confusing aspect is the powder factor.

When confronted with vibration problems, some engineers propose to reduce the powder factor of the blast, but nothing is farther from the minimum level situation. Blasts have been recorded in which the powder factor was reduced 20% from the optimum and the vibration levels measured were two or three times higher as a consequence of the confinement and poor spatial distribution of the explosive, causing lack of displacement and swelling energy.

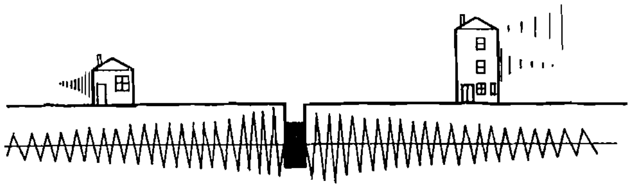
In Fig. 33.5, the powder factor influence can be observed in extreme situations and close to the optimum level in bench blasting.



a. EFFECT OF DISTANCE



b. EFFECT OF GEOLOGY



c. EFFECT OF RECEIVING STRUCTURES

Fig. 33.4. Effects of site conditions on blast vibrations.

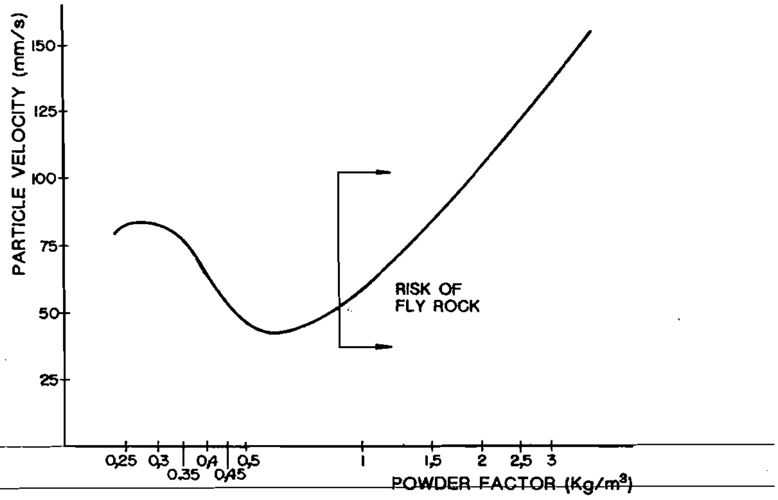


Fig. 33.5. Powder factor influence on vibration intensity.

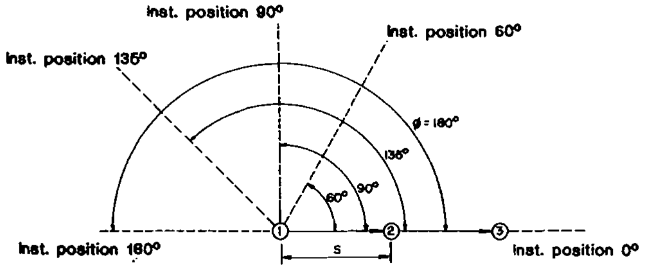


Fig. 33.6. Typical recording instrument positions and firing angles for a 3 hole blast, with a firing sequence of 1-2-3 from left to right (Wiss and Linehan).

33.2.5 *Types of explosives*

There is a correspondence between the particle velocities and the strains induced in the rocks, and this constant of proportionality is the impedance of the rock medium.

Therefore, the first practical consequence is that those explosives which generate lower blasthole pressures will also produce lower vibration levels. These explosives are those of low density and detonation velocity such as ANFO. If the same amount of ANFO is compared with a common slurry, or with an aluminized watergel, the intensity of vibrations generated by the first is 2 and 2.4 times lower respectively. This finding has been supported by various engineers such as Hagan and Kennedy (1981), Matheu (1984), etc.

In vibrographic studies, if explosives of very different strengths are used, the charges should be normalized to a standard explosive of known strength. Usually ANFO is chosen as the reference explosive, as it is the most widely used.

33.2.6 *Delay Period*

The delay intervals between blasthole detonations can be referred to as the nominal delay or effective delay time.

The first is the difference between the nominal initiation times, while the effective delay time is the difference of the arrival times of the pulses generated by blasthole

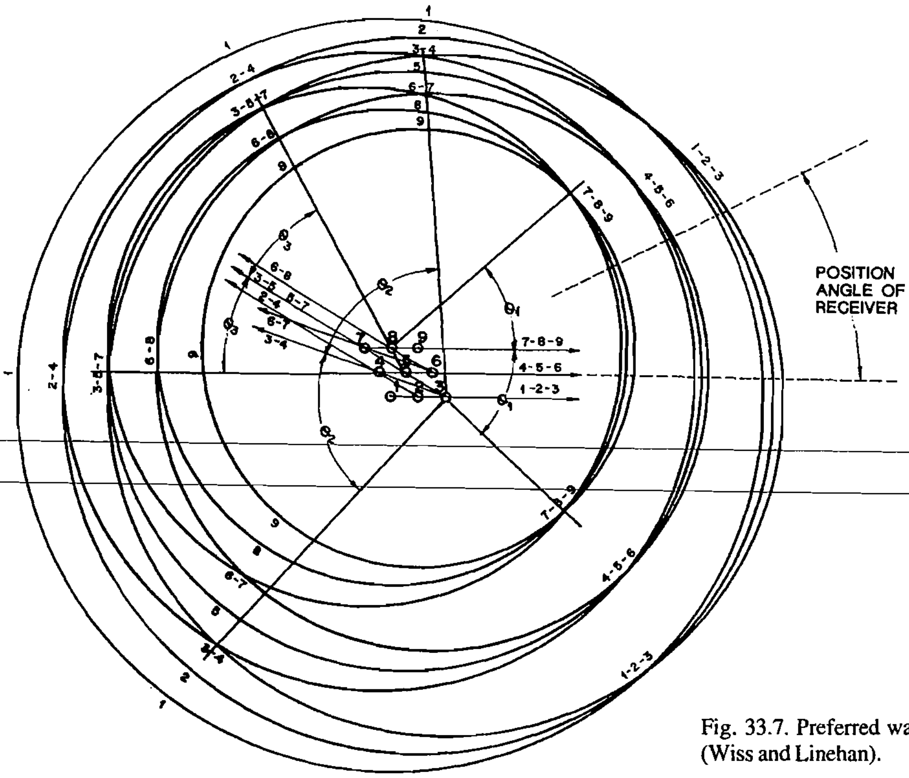


Fig. 33.7. Preferred wave collaboration directions in a multiple blast (Wiss and Linehan).

detonation fired with consecutive periods. In the simple case of a single row of holes, these parameters are interrelated by the following equation:

$$t_e = t_n - \frac{S \times \cos \phi}{VC}$$

where: t_e = Effective delay time, t_n = Nominal delay time, S = Spacing between holes, VC = Propagation velocity of the seismic waves, ϕ = Angle between successively detonated holes and the position of the sensor or recording instrument.

In Fig. 33.6, the case of a single row of blastholes with different relative positions of the recording instruments.

The critical angle of the relative position where the seismic waves arrive at the same time and, therefore, a collaboration can occur between them, will be that where $t_e = 0$, and can be determined from:

$$\phi_c = \arccos \frac{VC \times t_n}{S}$$

In Fig. 33.7, a multiple blast is represented and the directions where there is a more probable interaction of the waves according to the theoretical break direction of the holes.

When referring to the minimum delay time that eliminates constructive interferences or has summing or interacting effects, in the first studies carried out by Duvall et al. (1963), intervals of 8 and 9 ms were suggested, calculated from the testing done in limestone quarries. Langefors (1963) points out that with intervals of more than 3 times the vibration period it can be assumed that there is no interaction between adjacent blastholes that are detonated in sequence, as the signals are absorbed. Wiss and Linehan (1978) suggest a nominal delay time between successive delay intervals of 17 ms, to eliminate the summing effect of the vibrations. In another study done by Nobel's Explosives Co. of Great Britain, on secuenced blasts with delay times between charge weights per hole of under 25 ms, the existance of constructive interferences in the maximum vibration level is confirmed, Fig. 33.8.

33.2.7 *Geometric parameters of the blasts*

The majority of the geometric design parameters have a considerable influence on vibrations generated by blasting. Some comments on the subject are:

– *Drilling diameter.* The increase in drilling diameter is negative as the amount of explosive per hole is proportional to the square of the diameter, which would give very high charge weights per hole on occasions.

– *Bench height.* The relationship $H/B > 2$ should be maintained, whenever possible, in order to obtain good fragmentation and eliminate toe problems, as well as reducing vibration levels because the charges are less confined.

– *Burden and spacing.* If the burden is excessive, the explosion gases find resistance to fragmentation and rock displacement, and part of the explosive energy is transformed into seismic energy which increases vibration intensity, Fig. 33.9. This phenomenon is most noticeable in presplitting blasts, where total confinement exists and vibrations of around five times those of a conventional bench blast can be registered.

If burden size is small, the gases escape and expand towards the free face at a very high speed, giving impulse

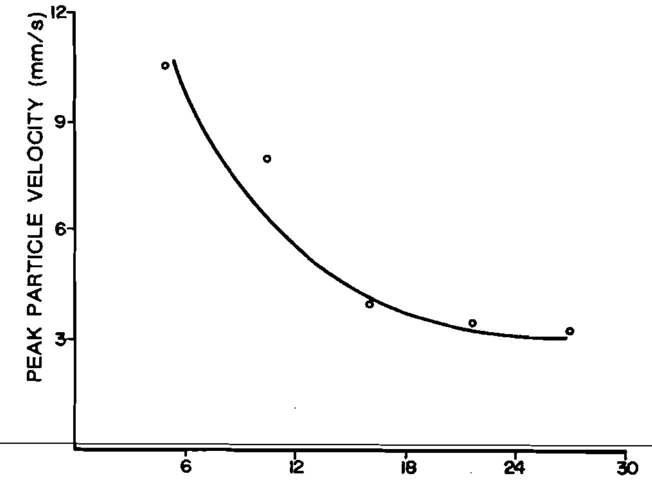


Fig. 33.8. Influence of the delay period upon the maximum vibration level.

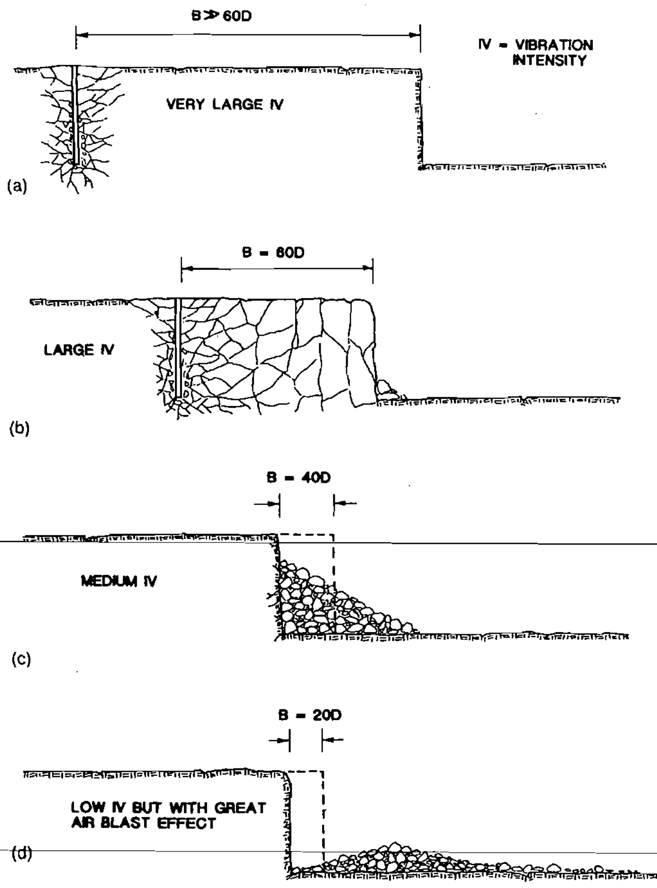


Fig. 33.9. Effects of the explosion according to the distance of the charged shothole from the free face (Berta, 1985).

to the rock fragments and projecting them uncontrollably, apart from provoking an increase in air blast and noise.

Spacing has a similar influence and its dimension actually depends on the burden value.

– *Subdrilling.* When longer than necessary lengths are used, each additional section collaborates each time with a lesser amount of energy for shearing and rock movement at the base, which means that a higher percentage of the explosive energy is converted into ground vibrations. This also makes for superfluous expense in drilling and explosives, and the floor is left irregular.

– *Stemming.* If stemming is too high, apart from fragmentation problems, confinement is increased and vibration levels are possibly higher.

– *Blasthole inclination.* Inclined blastholes allow better use of energy at floor level, and even reduce vibrations.

– *Decked charges (decoupling).* Tests carried out by Melnikov, using decked charges of 65 to 75%, show that fragmentation is improved and the size distribution is more uniform.

The percentage of secondary blasting is reduced from 2 to 10 times as well as the powder factor and the intensity of ground vibrations, Fig. 33.10.

– *Size of the blasts.* The dimensions of the blasts are limited, on one hand, by the maximum charge weights per hole that have been determined in the vibrographic studies based on the laws of propagation, types of structures to be protected and characteristic parameters of the disturbance phenomena.

33.3 CHARACTERISTICS OF GROUND VIBRATIONS

In the following paragraphs some theoretical aspects of the generation and propagation of vibrations produced in rock blasting are analyzed; although it must be indicated that this is just a mere approximation to the problem, as the actual phenomena are much more complex owing to the interaction of different types of waves and their modifying mechanisms.

33.3.1 *Types of generated seismic waves*

The vibrations generated in blasting are transmitted through the ground as seismic waves. The wave front is displaced radially from the point of detonation. The different seismic waves are classified in two groups: *body waves* and *surface waves.*

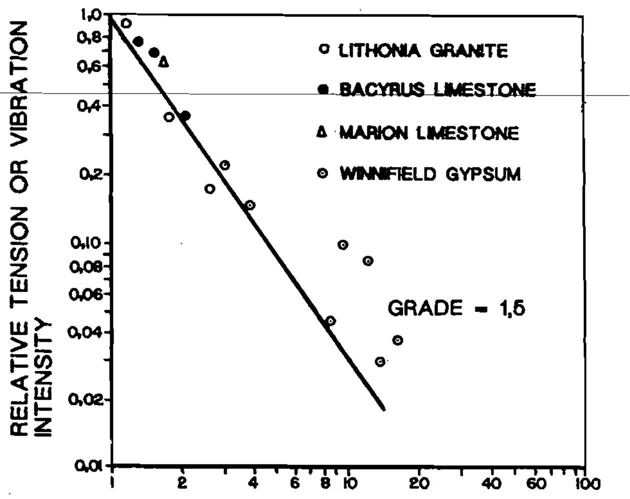


Fig. 33.10. Decked charge influence in vibration intensity.

The first type of body waves are called *Primary or Compressional.* These wave propagate through the ground materials alternatingly producing compressions and dialations, with particle movement in the direction of wave propagation. These are the fastest waves and they change the volume but not the shape of the materials through which they propagate.

The second type are made up of *Transverse or Shear – S waves* which move the particles in a direction that is perpendicular to that of wave propagation, Fig. 33.11.

The velocity of the transverse waves is somewhere between that of the longitudinal waves and the surface waves. The materials through which they propagate change in shape but not in volume.

The surface waves that are usually generated in rock blasts are: Rayleigh-R waves and Love-Q waves. Other types of surface waves are the Channel waves and the Stonelly waves which are not important as they supply very little information.

The Rayleigh waves are characterized by eliptical particle orbit, usually a motion that is contrary to the propagation direction of the wave. The Love waves are faster than the Rayleigh and give particle motion that is transverse to that of propagation.

The propagation velocity of the *P* and *S* waves depends on the elastic constants of ground materials and can be estimated from the following equations:

$$VC_p = \sqrt{\frac{E \times (1 - \nu)}{\rho_r \times (1 - 2\nu) \times (1 + \nu)}}$$

$$VC_S = \sqrt{\frac{E}{2 \times \rho_r \times (1 + \nu)}}$$

where: ρ_r = Rock density, ν = Poisson's ratio, E = Young's modulus, VC_p and VC_s = Propagation velocities of the longitudinal and transverse waves, respectively.

For a material with a Poisson coefficient of 0.25, it can be stated that VC_p is 1.73 times VC_s, and that the velocity of the Rayleigh waves is $0.9\ VC_s$.

As the waves travel with different velocities and the the number of delays in the blasts can be large, the generated waves interact with one another in time and space, producing for complex movements which require that the instruments be placed in three directions: radial, vertical and transversal, Fig. 33.12.

The distribution of the energy transported by the different types of waves has been studied by several investigators such as Miller and Pursey (1955), Vorob'ev (1973), etc. who have come to the conclusion that the Rayleigh waves carry between 70 and 80% of the total energy.

In the blasting manual by Du Pont, it is stated that this type of wave dominate the surface ground movement at several hundred meters from the blast and, as many structures and buildings around the operations are farther than 500 m away, the Rayleigh waves constitute the highest potential damage risk.

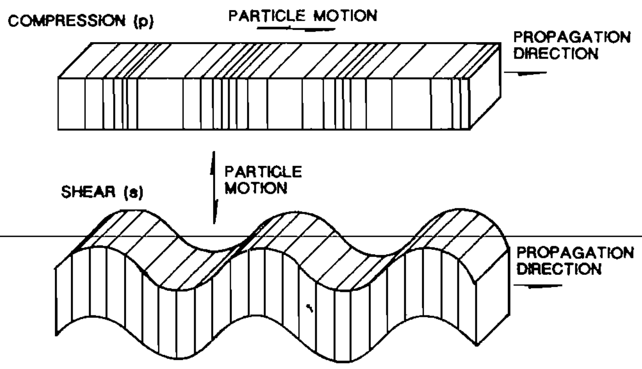


Fig. 33.11. Compressive-P and Shear-S waves.

33.3.2 *Wave parameters*

The passing of a seismic wave through a rock medium puts all of its particles in motion, which is called vibration.

A simplification for the study of blast generated vibra-

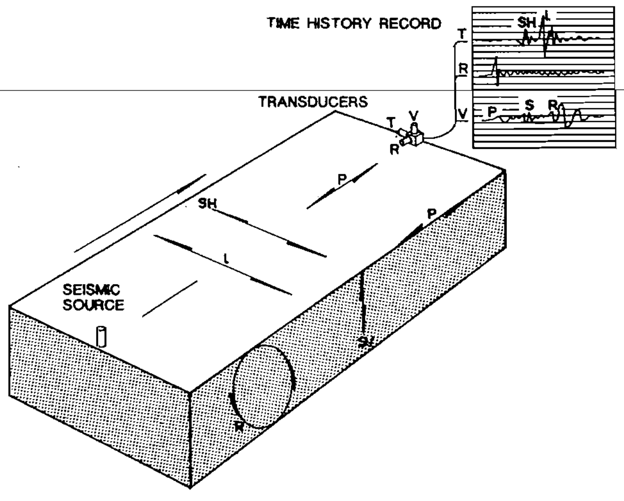


Fig. 33.12. Different wave types.

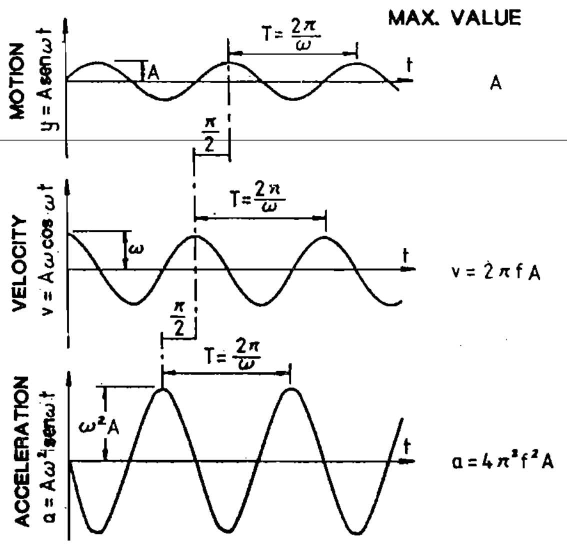


Fig. 33.13. Harmonic wave motion.

tions consists of considering these as harmonic motion type waves, Fig. 33.13.

The basic parameters for analysis are:

- *Amplitude (A)*. Maximum displacement of a particle from its rest position.
- *Particle velocity (v)*. Velocity at which a particle moves.
- *Acceleration (a)*. Velocity per unit time, i.e., $a = v/t$.
- *Frequency (f)*. Complete number of oscillations or cycles per second. The frequency is the inverse of the period T_s.

The displacement y at any instant is worth:

$$y = A \times \text{sen}\,(\omega t)$$

where:

$$\omega = 2 \times \pi \times f = 2 \times \pi \times \left(\frac{1}{T_s}\right)$$

The length of the wave λ for a propagation velocity of VC is:

$$\lambda = VC \times T_s = VC \times \left(\frac{1}{f}\right)$$

The relationships between displacement, velocity and acceleration of the particle are:

$$y = A \times \text{sen}\,(\omega t)$$

$$v = \frac{\mathrm{d}y}{\mathrm{d}t} = A \times \omega \times \cos\,(\omega t)$$

$$a = \frac{\mathrm{d}v}{\mathrm{d}t} = -\,A \times \omega^2 \times \text{sen}\,(\omega t)$$

When only the maximum absolute values of these parameters are taken into account, the previous relationships are converted into:

$$v_{\max} = A \times \omega = A \times 2 \times \pi \times f$$

$$a_{\max} = A \times \omega^2 = A \times 4 \times \pi^2 \times f^2 = v_{\max} \times 2 \times \pi \times f$$

33.3.3 *Geometric attenuation*

The density of the energy of propagating waves generated by the detonation of an explosive charge diminishes as the waves reach larger volumes of rock. Given that the ground vibrations induced by the blasts comprehend a complex combination of waves, it would seen logical to take into consideration certain geometric attenuation factors for each type. In a homogeneous, elastic and isotropic medium, the amplitude drops due to geometric absorption, and its drop, for different types of dominating waves, is proportional to:

- $1/DS$ for body waves in an (semi)infinite medium.
- $1/DS^{0.5}$ for Rayleigh waves.
- $1/DS^2$ for body waves that travel along a free surface.

Where DS is the distance from the seismic source (Richart et al. 1970).

33.3.4 *Non-elastic absorption*

In nature, the rock masses do not constitute an elastic, isotropic and homogeneous medium for vibration propagation. To the contrary, numerous non-elastic or non-dispersive effects appear which provoke a loss of energy during wave propagation, which is added to that caused by geometric attenuation. There are numerous reasons for the non-elastic attenuations, and each has different degrees of influence:

- Dissipation in a nonelastic matrix owing to the relative movement in the intercrystaline surfaces and planes of discontinuity.
- Attenuation in saturated rocks owing to fluid movement with respect to the matrix.
- Flow inside the cracks.
- Dispersion of stresses induced by absorbed volatiles.
- Reflection in porous rock or with large cavities.
- Energy absorption in systems that have phase changes, etc.

33.3.5 *Interaction of elastic waves*

The interaction of seismic waves in time and space can bring about a concentratin or focusing which gives attenuation coefficient values that are higher or lower than predicted or theoretically calculated.

The topography and geometry of the geological formations can produce the reflection and concentration of wave fronts in certain points.

33.4 AIR BLAST CHARACTERISTICS

Air blast is the pressure wave that is associated with the detonation of an explosive charge, whereas noise is the audible and infrasonic part of the spectrum: from 20 Hz to 20 kHz. Air blasts are the low frequency air vibrations with values that are usually under 20 Hz.

According to Wiss and Linehan (1978), the causes of these disturbances are the following:

1. Ground vibration brought on by an explosion (Rock pressure pulse).
2. Escape of gases from the blasthole when the stemming is ejected (Stemming release pulse).
3. Escape of gases through the fractures created in the rock mass face (Gas release pulse).
4. Detonation of the initiating cord in the open air.
5. Displacement of the rock at bench face as the blast progresses (Air pressure pulse).
6. Collision between the projected fragments, Fig. 33.14.

The combination of vibrations associated with these sources give a mobile front of air overpressure that travels from the blast point. As air is compressible, it absorbs part of the pressure wave energy to later set it free

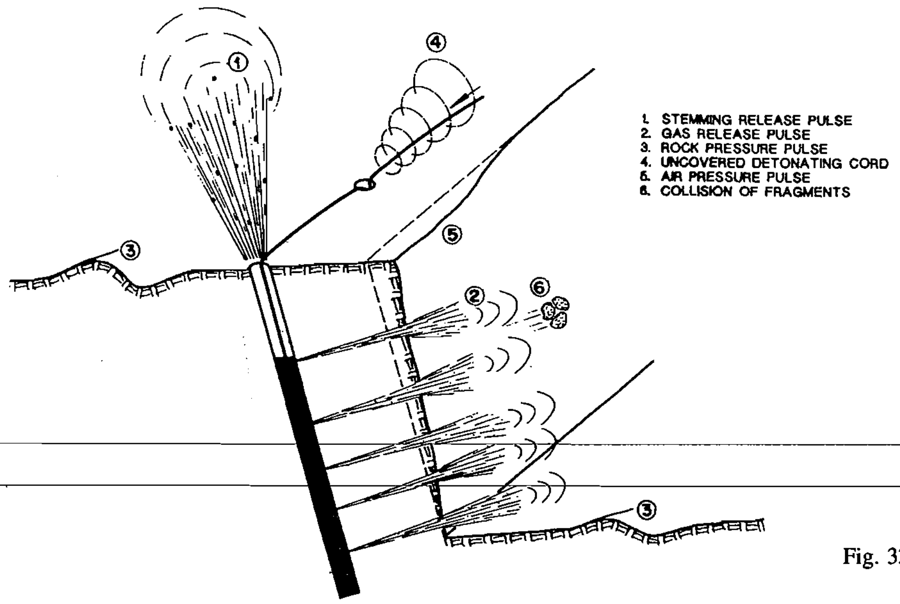

①
④
1. STEMMING RELEASE PULSE
2. GAS RELEASE PULSE
3. ROCK PRESSURE PULSE
4. UNCOVERED DETONATING CORD
5. AIR PRESSURE PULSE
6. COLLISION OF FRAGMENTS
③
⑤
②
⑥
③

Fig. 33.14. Air blast fronts in blasting.

Photo 33.2. Effect of a detonating cord covered with sand.

through expansion of the hot gases, causing depression in those points.

Air blast characteristics are not easy to predict. Factors such as climate, topography, etc. intervene which, along with the actual blast design, can give different results in each case.

As mentioned before, air blast contains a considerable amount of low frequency energy which can eventually produce direct damage on structures; however, high frequency vibrations are more common and are felt in windows, dishes, doors, etc.

33.5 INSTRUMENTATION FOR RECORDING AND ANALYZING VIBRATIONS AND AIR BLAST

In order to carry out a study of vibrations and air blast, special instrumentation is required, as follows:

- A seismograph system which detects and records ground movement.
- A computer system which analyzes the recorded signals.

The most frequent ranges of the different characteristic parameters of blast induced vibrations are shown in Table 33.2.

33.5.1 *Recording and analyzing equipment*

The recording system consists of several components which carry out the following functions:

- Detection by sensors.

Table 33.2.

Parameter	Range
Displacement	10^{-4} to 10 mm
Particle velocity	10^{-4} to 10^{3} mm/s
Particle acceleration	10 to 10^{5} mm/s
Length of pulse	0.5 to 2 s
Wave length	30 to 1500 m
Frequency	0.5 to 100 Hz

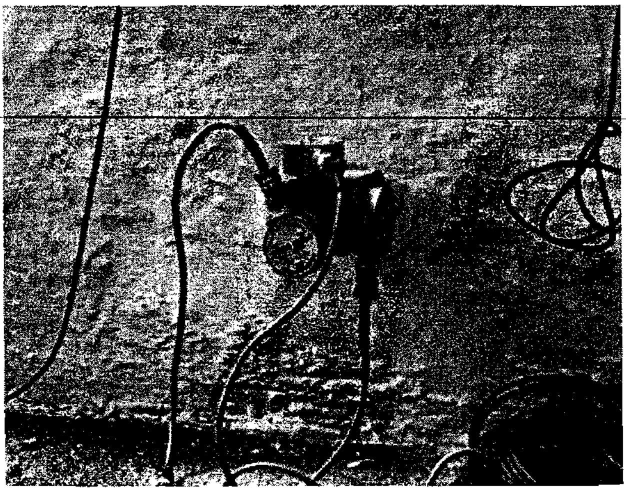


Photo 33.3. Triaxial recording station consisting of three accelerometers.

– The transmission of the electric signals emitted by the sensors through the conductor wires, and

– Recording of the signals with a seismograph for its posterior study and analysis.

In Fig. 33.15, a schematic diagram of the operations and instrumentation used in the study of vibrations is represented.

The sensors make up the first element of the measuring system and should be planted well in contact with the ground so that they vibrate as part of the earth, emmiting the signal which represents the true ground motion. This contact can be achieved by simply placing the sensors on the ground, by screwing them to blocks of aluminum or other nonferric material, which is the least recommendable if the sensors are electrodynamic; anchoring them next to a metal block by means of an expansion plug introduced into a hole made in the rock, which is the most common system used in firm ground; setting the sensors inside a box and burying it in the earth, used when the ground is not consolidated; and other less frequent alternatives such as drilling blastholes and cementing them to create a firm base, using synthetic resins, etc.

There are two tendencies as to where the sensors should be placed: one, on the ground near the structures to be protected; and another, on the structures themselves, bearing in mind that the latter will reflect the response of the construction and not record the ground movement.

As to the vibration sensors, the most widely used are the vibration seismographs and the acceleration seismographs. The first are the most popular, as particle velocity has become the parameter used to correlate the vibrations with the damage produced by the blasts.

They are electromagnetic type transducers which emit an electric tension that is proportional to the velocity of the vibration particle. The electric signal is generated by a mobile coil within the field of a stationary magnet, Fig. 33.16. The range of application is limited by the actual resonance frequency of the vibration seismograph, which is usually between 5 and 15 Hz and up to a maximum of 200 to 300 Hz. As can be seen, they are not recommendable when low frequencies exist.

The acceleration seismographs are based on the difference of potential generated by a piezoelectric crystal under force. This force is proportional to the mass of the cristal by the acceleration of vibratory movement, Fig. 33.17.

The recorders are instruments that allow visualization and amplification of the signals coming from the sensors. They can be of various types: those which only record peak values on paper, those which are continuous on photographic paper, printed by ultraviolet light galvanometers, those of needle and thermic paper, those which use magnetic tape cassette or record, recording analogic signals registered by the sensors.

These have the advantage of allowing the signal to be reproduced whenever necessary, introducing filters, integrators, etc. between said signal and the recorder.

The analogic recording on magnetic tape is carried out with different techniques which are adapted to the pertinent conditions: modulated frequency recording – inter-

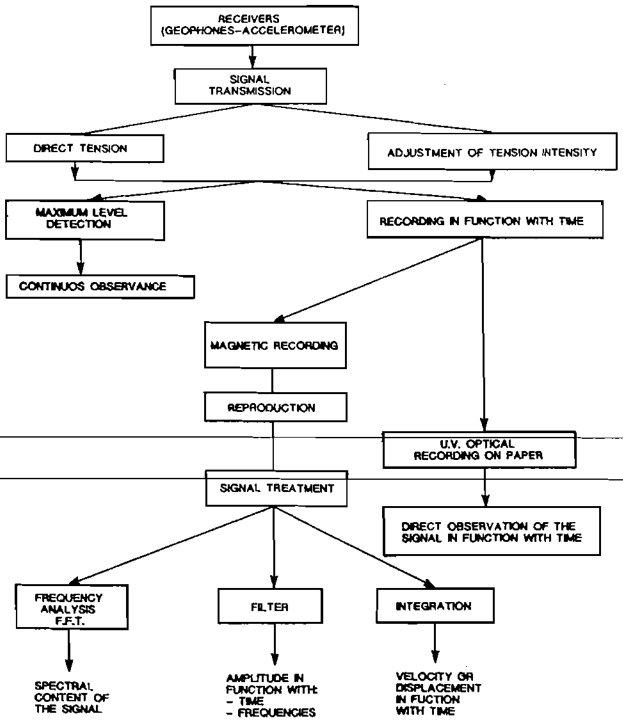

RECEIVERS (GEOPHONES–ACCELEROMETER)
SIGNAL TRANSMISSION
DIRECT TENSION
ADJUSTMENT OF TENSION INTENSITY
MAXIMUM LEVEL DETECTION
RECORDING IN FUNCTION WITH TIME
CONTINUOS OBSERVANCE
MAGNETIC RECORDING
REPRODUCTION
U.V. OPTICAL RECORDING ON PAPER
SIGNAL TREATMENT
DIRECT OBSERVATION OF THE SIGNAL IN FUNCTION WITH TIME
FREQUENCY ANALYSIS F.F.T.
FILTER
INTEGRATION
SPECTRAL CONTENT OF THE SIGNAL
AMPLITUDE IN FUNCTION WITH: - TIME - FREQUENCIES
VELOCITY OR DISPLACEMENT IN FUCTION WITH TIME

Fig. 33.15. Schematic diagram of the recording and analysis of vibrations.

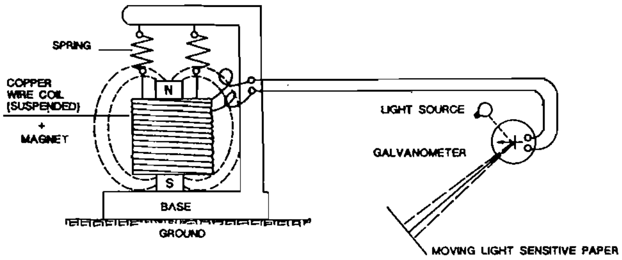

SPRING
COPPER WIRE COIL (SUSPENDED) + MAGNET
N
S
BASE
GROUND
LIGHT SOURCE
GALVANOMETER
MOVING LIGHT SENSITIVE PAPER

Fig. 33.16. Velocity gauge.

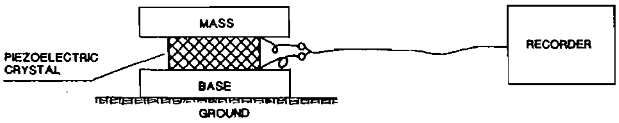


Fig. 33.17. Acceleration seismograph (accelerometer).

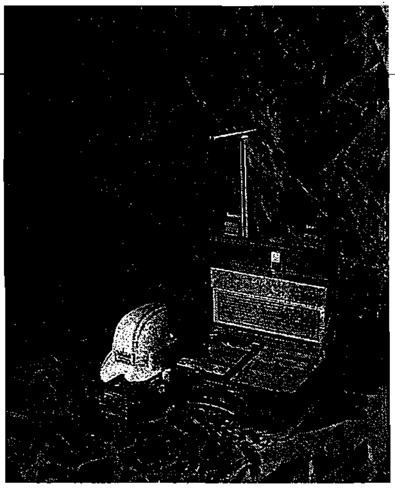


Photo 33.4. Seismograph for blast monitoring.

esting for low frequencies, direct recording for high frequencies, and multiplexed recording when a large number of signals come in.

The seismograph system is usually composed of analog or digital instruments to reproduce and visualize the signals.

When the signals are recorded on magnetic tape, these can be reproduced for a complete analysis, including the calculation of the Rapid Transform of Fourier in order to obtain the density of impulse frequency received or the energy destribution of seismic movement as function of the frequency. Apart from this, with the graphics obtained as function of the time, the maximum vibration level and its corresponding period can be predicted, as well as the length of the disturbance, etc.

When necessary, the signals can be filtered, integrated or derived, in order to eliminate certain components or calculate other parameters from the primitive recording; for example if acceleration has been measured, integrate one or two times to obtain particle velocity or displacement, respectively.

Lastly, it should be indicated that the sensors, although treated with care, should be checked periodically for sensitivity, and possible variation with use.

Air blast is usually measured with a sonometer, which is easy to transport and install. It should be placed away from reflecting surfaces, in front of shielding objects and making certain that there is no background noise or wind to modify the recording.

Special attention should be paid in selecting the scale of consideration, according to the measurements required.

33.6 CALCULATORS OF PROPOGATION LAWS FOR LAND AND AIR VIBRATIONS

One of the fundamental stages in the study and control of vibrations generated by blasting is the determination of the laws that govern their propagation in different mediums of land or air.

There are several methods used to estimate the ground movements produced by blasting. These methods are relatively simple as, if not, they would not have been readily accepted in the practical field of mining and civil engineering.

33.6.1 *Calculators for ground vibrations*

One of the first propagation equations was suggested by Morris (1950) and is as follows:

$$A = K \times \frac{\sqrt{Q}}{DS}$$

where: A = Maximum particle amplitude (mm), Q = Explosive charge weight (kg), DS = Distance from blast to recording point (m), K = Characteristic constant of the site which varies from 0.57, for competent hard rocks, up to 3.40 for unconsolidated ground.

Leconte (1967), when revising the vibration control techniques suggested substituting the maximum particle amplitude of the Morris equation for the vector sum of the particle velocity, as follows:

$$v = K_{vr} \times \frac{\sqrt{Q}}{DS}$$

Amongst the most rigorous posterior investigations, those of Blair and Duvall (1954) and Duvall and Petkof (1959) are worthy of mention as they also try to correlate the intensity of generated seismic movement with the explosive charge weight and the distance to the source. In the supposition that the explosive column is a symmetrical sphere, the conclusion was that any lineal dimension should be corrected by the cubic root of the explosive charge. Similar results were obtained by Ambraseys and Hendron (1968) and Dowding (1971).

In a general sense and taking particle velocity as the most characteristic vibration parameter, it was found that the intensity of the seismic waves and the scaled distance (cocient between the distance and the charge elevated to an exponent) followed the law below:

$$v = K \times \left[\frac{DS}{Q^{1/3}}\right]^{-n}$$

where: v = Particle velocity, DS = Distance, Q = Maximum charge per delay, K, n = Empirical constant.

If cylindrical charges are used, it has been observed by dimensional analysis that the distances should be corrected by dividing them by the square root of the charge, Devine (1962), Devine and Duvall (1963), then being able to define the following laws of propagation, Fig. 33.19:

$$v = K \times \left[\frac{DS}{Q^{1/2}}\right]^{-n}$$

This formula has been one of the most widely used up to present by numerous investigators, official organisms, users and manufacturers of explosives.

Other authors such as Atewel et al. (1965), Holmberg and Persson (1978), and Shoop and Daemen (1963) do not take into consideration a particular charge symmetry and use the following general equation:

$$v = K \times Q^{a} \times DS^{b}$$

where K, a and b are empirical constants estimated for a determined site by means of a multiple regression analysis.

At relatively small distances, in comparison with the charge length, the propagation law $v = K \times Q^{a} \times DS^{b}$ can be modified by taking into account the following geometric model, Fig. 33.20.

If one takes as basis a lineal charge concentration q_l (kg/m), the particle velocity v can be obtained by integrating the previous equation with respect to the relative position along the length of the charge.

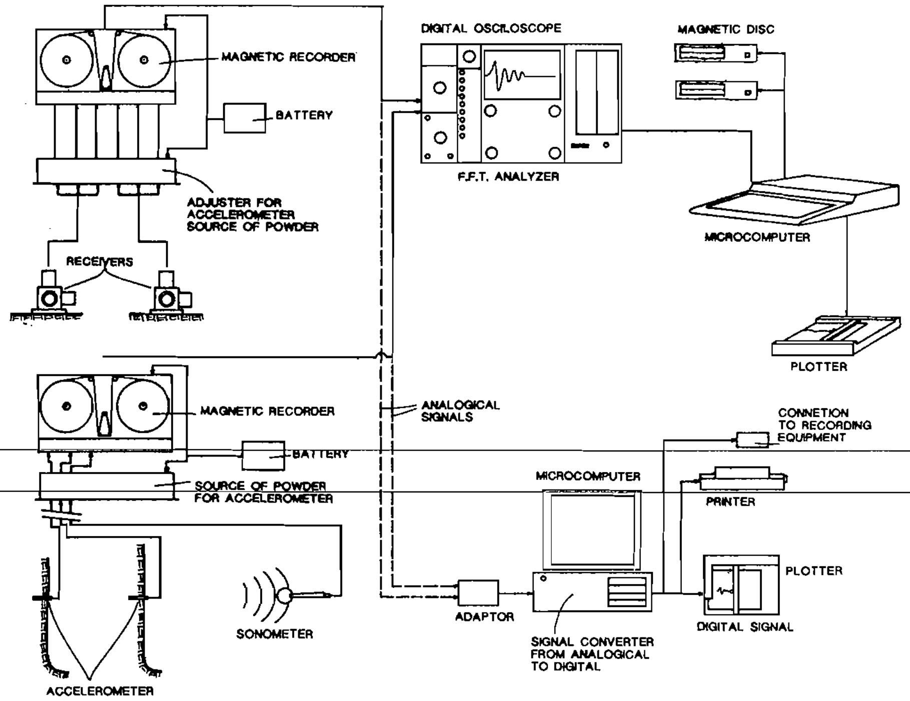


Fig. 33.18. Seismograph systems for recording and analyzing vibrations and air blast.

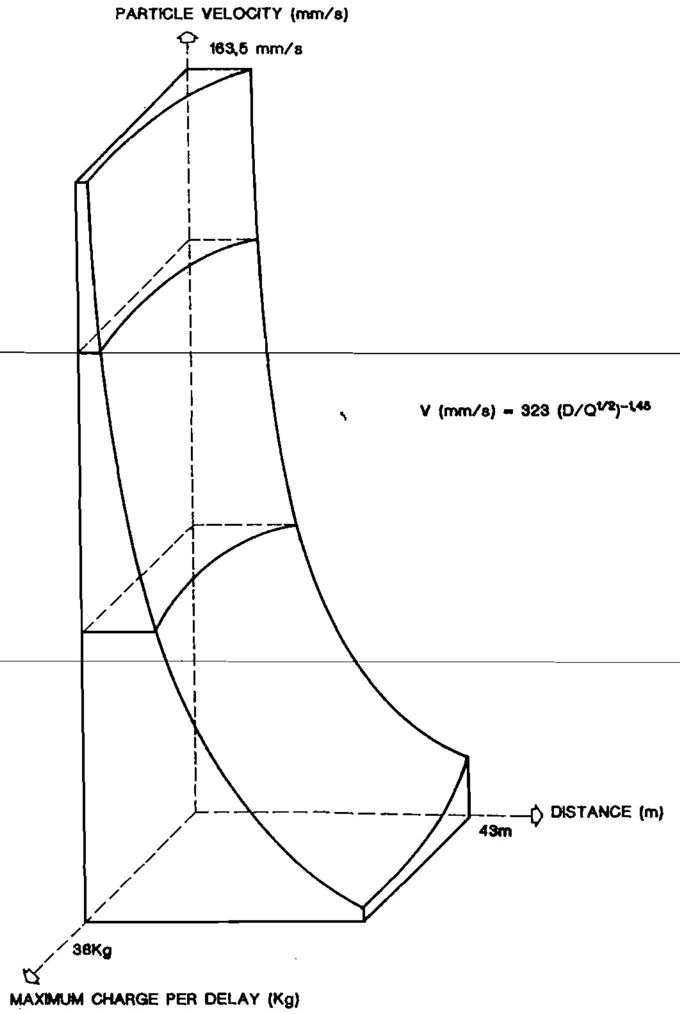


Fig. 33.19. Tridimensional representation of a vibration propagation law.

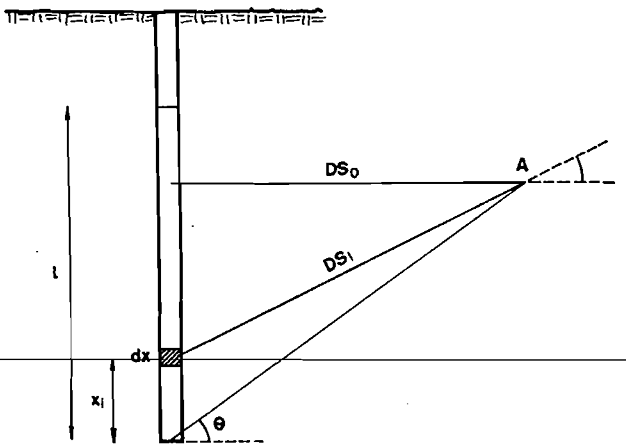


Fig. 33.20. Integration over charge length to calculate particle velocity at an arbitrary observation point (Holmsberg and Persson).

The distance from any part of the charge to point *A* is given by:

$$DS_i^2 = DS_o^2 \times (DS_o \times \text{tag}\,\theta - x_i)^2$$

where: DS_o = Minimum distance from charge to point A, θ = Angle of inclination, x_i = Distance from lower end of the elemental charge q_i.

$$q_i = q_l \times \mathrm{d}x$$

Integrating l along the total length of the charge, the maximum particle velocity is given by:

$$v = k \times q_1{}^a \times \left[\int_o^1 \frac{\mathrm{d}x}{DS_o^1 + (DS_o \times \text{tag}\,\theta - x)^{2\,b/2a}} \right]^a$$

For competent rocks, such as Swedish granites, there are some constants with values $k = 700$, $a = 0.7$ and $b = -1.5$, with v expressed in mm/s.

In the Figs 33.21 and 33.22, the value of v is shown as function of *DS*, minimum distance from the point of interest to the elongated charge, and the lineal charge concentration for an explosive such as ANFO.

This method of calculation is very interesting when wishing to preserve the resistance characteristics of the remaining masses, in surface mining slopes as well as underground walls, as it enables the estimation of maximum charges for blastholes near the surface of the cut.

The Swedish school, headed amongst others by Langefors, Kilhström and Gustafsson, relates the charge levels $Q/DS^{3/2}$ with particle velocity by using the equation:

$$v = K \times \left[\frac{Q}{DS^{3/2}}\right]^{1/2}$$

Lundborg (1977), basing his observations on data of the US Bureau of Mining (Nicholls et al, 1971) found a law $v = f(DS, Q)$, and proposed the following equation:

$$\log v = 4.08 + 0.14 \log Q - 2.06 \log DS + 0.22 \log Q \times \log DS$$

which is represented as a tridimensional surface. A simplification consists in adapting a plane to said surface, obtaining the following equation:

$$\log v = 2.86 + 0.66 \log Q - 1.54 \log DS$$

The investigations carried out in the last few years have permitted a better prediction than with the typical lines represented on bilogarithmic paper, using the curved lines in correlations following the tendencies of the pairs of data Just and Free (1980), and López Jimeno et al. (1985). Although the exponential fall has been acknowledged for some time, e.g. Duvall and Petkoj (1959), it has not been taken into account in the predictor equations until recently.

Just and Free (1980), propose the following propagation law, based on observations in controlled blasts:

$$v = K \times (DS/Q^{1/3})^{-1} \times e^{-\alpha(DS/Q^{1/3})}$$

assuming that the body waves are predominant and that spherical divergency exists.

Ghosh and Daemen (1983) take into consideration the nonelastic absorption to take into account the exponential fall of v, making it proportional to $e^{-\alpha DS}$, Fig. 33.23.

They suggest, depending upon the types of waves, the following propagation laws:

1. *Body waves* that are predominant (e.g. close to the blast) and measured on the surface:

$$v \,\alpha \frac{1}{(DR)^2} = \left[\frac{DS}{Q^{1/2}}\right]^{-2} \quad \text{and} \quad v \,\alpha\, e^{-\alpha DS}$$

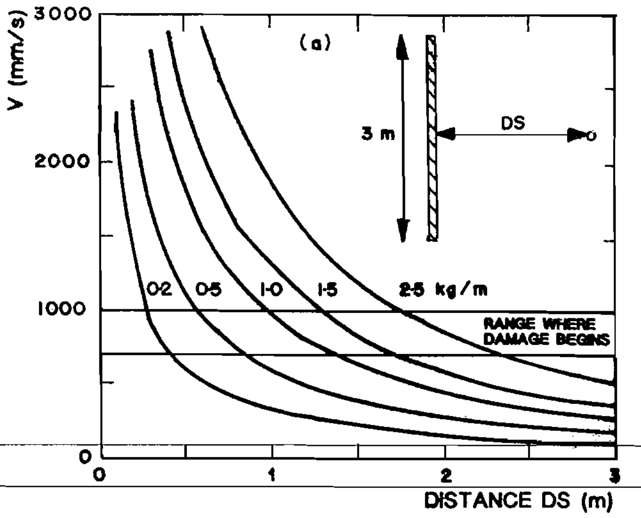


Fig. 33.21. Blastholes of small diameter and length charged with ANFO (Holmberg and Persson).

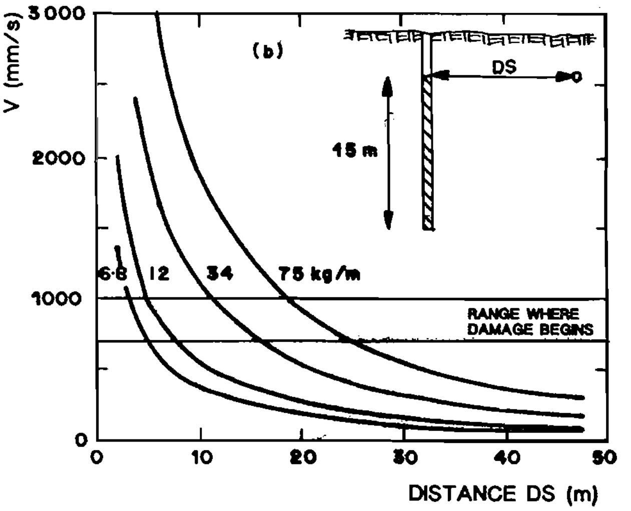


Fig. 33.22. Blastholes of large diameter and length charged with ANFO (Holmberg and Persson).

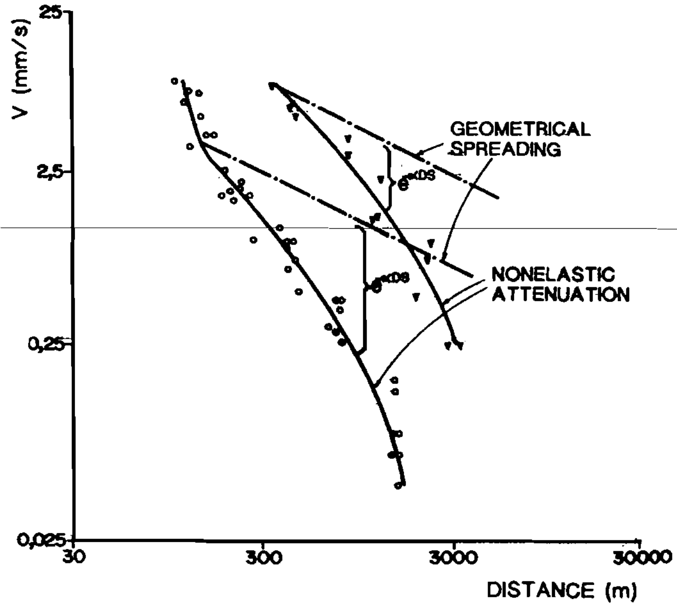


Fig. 33.23. Vibration propagation laws with geometric absorptions and nonelastic exponential attenuations (Ghosh and Daemen, 1983).

where DR = Scaled distance.

Therefore, the following exists:

$$v = K_1 \times \left[\frac{DS}{Q^{1/2}}\right]^{-2} \times e^{-\alpha DS}$$

2. *Body waves* that are predominant (e.g. close to blast) and measured under the ground surface.

$$v \,\alpha \frac{1}{(DR)} = \left[\frac{DS}{Q^{1/2}}\right]^{-1} \text{, therefore}$$

$$v = K_2 \times \left[\frac{DS}{Q^{1/2}}\right]^{-1} \times e^{-\alpha DS}$$

3. *Rayleigh waves* that are predominant (e.g. at large distances from the blast).

$$v \,\alpha \frac{DS}{(DR)^{0.5}} = \left[\frac{DS}{Q^{1/2}}\right]^{1/2} \text{, therefore}$$

$$v = K_3 \times \left[\frac{DS}{Q^{1/2}}\right]^{-1/2} \times e^{-\alpha DS}$$

The exponent of Q will depend upon the geometry of the explosive charge, as indicated previously, ⅓ for spherical charges and ½ for cylindrical. The general equations which enclose the former ones are, therefore:

$$v = K \times \left[\frac{DS}{Q^{1/2}}\right]^{-n} \times e^{-\alpha DS}$$

$$v = K \times \left[\frac{DS}{Q^{1/3}}\right]^{-n} \times e^{-\alpha DS}$$

33.6.2 *Theoretical prediction of ground vibrations*

When instrumentation and equipment to carry out a study of vibrations are available, the intensity of the disturbances originated by blasting can be predicted with a theoretical model, G. Berta (1985), taking into account that the seismic energy transmitted to the rock by the explosive can be evaluated with the two following equations:

$$E_s = 2\pi^2 A^2 f^2 \times 2\pi DS^2 \times \rho_r \times VC \times T_v \times 10^{-6}\,(MJ)$$

$$E_s = n_t \times n_1 \times n_2 \times E_T \times Q$$

where: A = Displacement (m), f = Frequency (Hz), DS = Distance from the explosion point (m), ρ_r = Density of the rock (kg/m^3), VC = Seismic velocity (m/s), T_v = Duration of the vibration (s), n_t = Breaking factor (Charges laid on the ground $n_t < 0.4$; Charges without a free face $n_t > 0.4$), n_1 = Impedance factor =

$$1 - \frac{(Z_e - Z_r)^2}{(Z_e + Z_r)^2}$$

n_2 = Coupling factor =

$$\frac{1}{e^{D/d} - 1{,}72}$$

E_T = Energy per unit of mass (MJ/kg), Q = Amount of explosive (kg), Z_e = Impedance of explosive (kg · m^{-2} · s^{-1}), Z_r = Impedance of rock (kg · m^{-2} · s^{-1}), D = Blasthole diameter (mm), d = Charge diameter (mm).

From the previous equations the following is obtained:

$$A(\mathrm{m}) = \sqrt{\frac{n_t \times n_1 \times n_2 \times E_T \times Q \times 10^6}{4 \times \pi^3 \times f^2 \times \rho_r \times VC \times DS^2 \times T_v}}$$

As the significative duration of vibration is considered to be five times the period:

$$T_v = 5\,T_s = \frac{5}{f}$$

and, as the ground frequency is calculated with:

$$f = (kf \times \log DS)^{-1}$$

where kf is a characteristic ground constant which influences the reduction of frequency with distance, Table 33.3. The amplitude and acceleration values can be calculated from:

$$A(\mathrm{m}) = \sqrt{\frac{n_t \times n_1 \times n_2 \times E_T \times Q \times kf \times \log DS \times 10^6}{20 \times \pi^3 \times \rho_r \times VC \times DS^2}}$$

$$v(\mathrm{m/s}) = \frac{\sqrt{Q}}{DS} \times \sqrt{\frac{n_t \times n_1 \times n_2 \times E_T \times 10^6}{5 \times kf \times \log DS \times \pi \times \rho_r \times VC}}$$

The previous formula is only valid when DS is over 1 meter.

Example

Consider a cylindrical charge of 10 kg in a granite bench with one free face.

The data of the explosive is:

E_T = 4.52 MJ/kg
$Z_e = 9.5 \times 10^6$ kg · m^{-2} · s^{-1}.

Table 33.3.

Type of ground	kf value
Water logged sands and gravel	0.11-0.13
Compact alluviums	0.06-0.09
Hard and compact rock	0.01-0.03

The characteristic rock parameters are:

$\rho_r = 2700\,\mathrm{kg/m^3}$
$VC = 5000\,\mathrm{m/s}$
$kf = 0.01$
$Z_r = 13.50 \times 10^6\mathrm{kg} \cdot \mathrm{m}^{-2} \cdot \mathrm{s}^{-1}$.

and the relationship blasthole diameter/charge diameter is $D/d = 1.06$.

What is the probable vibration intensity at a distance of 150 m?

$$v = \frac{\sqrt{10}}{150} \times \sqrt{\frac{0.4 \times 0.98 \times 0.85 \times 4.52 \times 10^6}{5 \times 0.01 \times \log 150 \times \pi \times 2700 \times 5000}} =$$

0.012 m/s = 12 mm/s

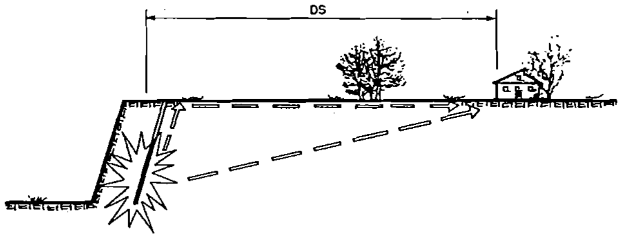


Fig. 33.24. Building situated at a distance DS from a position where blasting occurs.

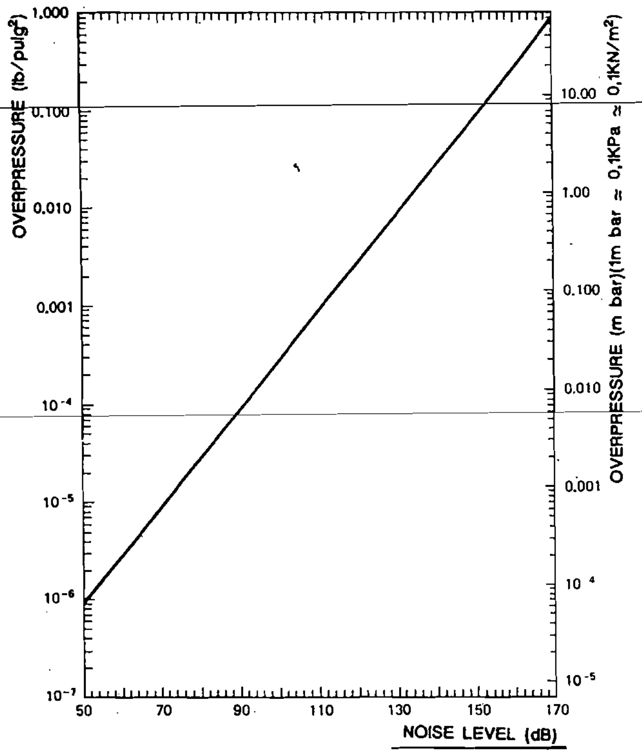


Fig. 33.25. Nomograph for overpressure conversion at noise level (Sisking et al. 1980).

33.6.3 *Air blast estimators*

The law of air blast propagation is accepted to be of the following type:

$$SP = K_1 \times \left[\frac{DS}{Q^{1/3}}\right]^{-K_2}$$

The audible component, which is the part of the spectrum comprehended be 20 Hz and 20 kHz, also called *noise*, is commonly measured in dB. The decibel is defined in terms of overpressure with the equation:

$$NR = 20 \log \frac{SP}{SP_o}$$

where: NR = Noise level, SP = Overpressure (N/m²), SP_o = Pressure of the lowest audible sound ($20 \cdot 10^{-6}$ N/m²), Figs 33.25 and 33.26.

If experimental data for air blast is not available, a first estimation can be found from the nomograph given by Ladegaard-Pedersen and Daily (1975), Fig. 33.27, obtained for bench blasting with a stemming height of 30*D*. Knowing the scaled distance and burden, the most probable air blast level can be determined.

33.7 STUDIES OF VIBRATION AND AIR BLAST

33.7.1 *Planning for study of vibrations*

The two basic objectives for a study of vibrations are:

– Finding the law of propagation of the vibrations to later determine the maximum charge weight per hole for a given distance and a previously adopted prevention criteria.

– Finding the predominating vibration frequencies for

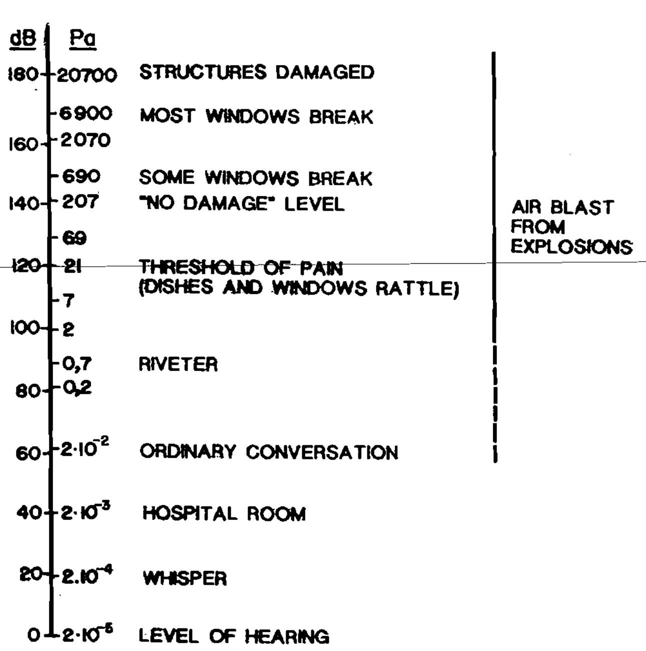


Fig. 33.26. Human and structural response to sound pressure level.

the rock mass to be excavated, thus permitting the most effective initiation sequence to be established.

To carry this out requires a previous geological analysis of the area between the blasts and the structures to be protected. In function with the findings, a scaled down blast can be designed, either individual or multiple, in which the charge weight per hole or distances can be varied, in order to cover a wide range of scaled distances.

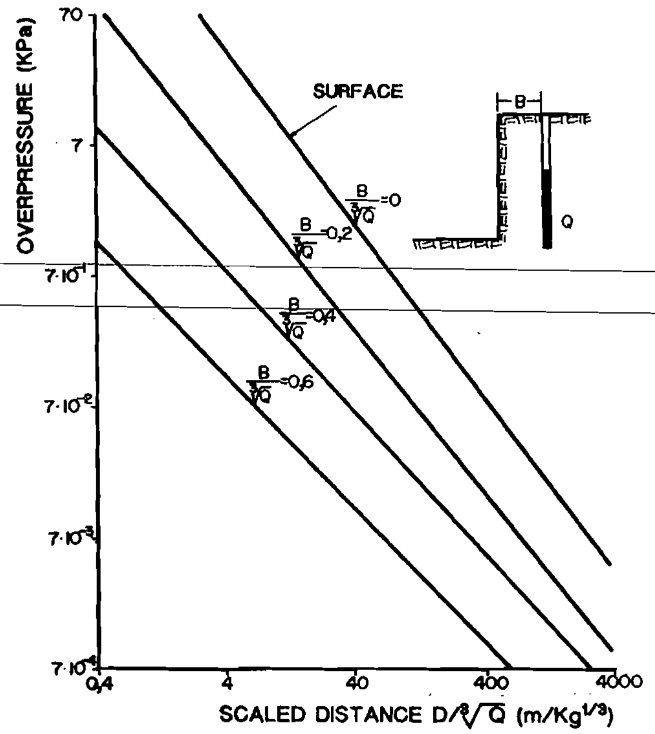


Fig. 33.27. Prediction of air overpressure from the geometry and charge of the blasts.

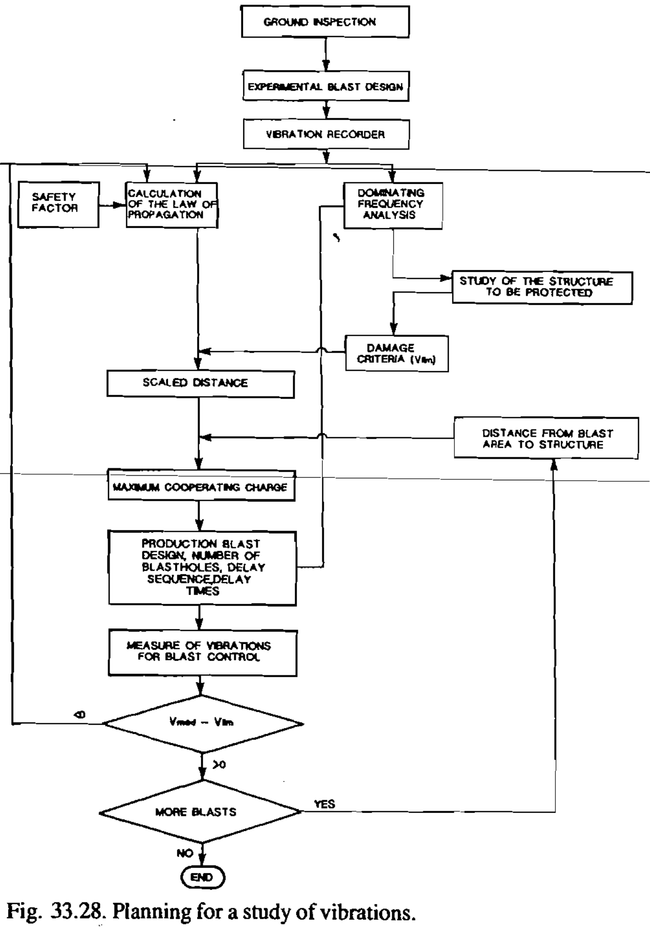

GROUND INSPECTION
EXPERIMENTAL BLAST DESIGN
VIBRATION RECORDER
SAFETY FACTOR
CALCULATION OF THE LAW OF PROPAGATION
DOMINATING FREQUENCY ANALYSIS
STUDY OF THE STRUCTURE TO BE PROTECTED
DAMAGE CRITERIA (Vm)
SCALED DISTANCE
DISTANCE FROM BLAST AREA TO STRUCTURE
MAXIMUM COOPERATING CHARGE
PRODUCTION BLAST DESIGN, NUMBER OF BLASTHOLES, DELAY SEQUENCE,DELAY TIMES
MEASURE OF VIBRATIONS FOR BLAST CONTROL
<0
Vmed − Vm
>0
MORE BLASTS
YES
NO
END

Fig. 33.28. Planning for a study of vibrations.

Once the results of the first blasts have been given, it can be decided which of the components is the most interesting to measure if the recording stations are not triaxial and, above all, when a large number of sensors are not available.

The minimum number of blasts recommended is between 8 and 10, and the execution conditions as to confinement, priming, etc. should be similar to those used in production blasting because frequently conservative postures are adopted, firing practically without a free face.

The spatial situation is also important, because a study carried out at a determined level and within a geological-structural context may not be, on occasions, extrapolated to other areas. All vibration studies have a limited value where space and time are concerned.

Once the records have been reproduced and analyzed in the laboratory, Fig. 33.29, they can be compared stadistically to ascertain the law of propagation.

Previously, all data will have been summed up in a Table, giving, for example, the maximum vibration levels, v if it is particle velocity, and the Scaled Distances DR, if the law to be obtained is of the following type:

$$y = a \times x^b$$

where: y = Particle velocity v, x = Scaled distance DR.

Logarithms can be taken and a straight line can be adjusted by squared minimums, Fig. 33.30.

$$\log y = \log a + b \times \log x$$

where:

$$b = \frac{\sum(\log x) \times (\log y) - \dfrac{(\sum \log x) \times (\sum \log y)}{n}}{\sum(\log x)^2 - \dfrac{(\sum \log x)^2}{n}}$$

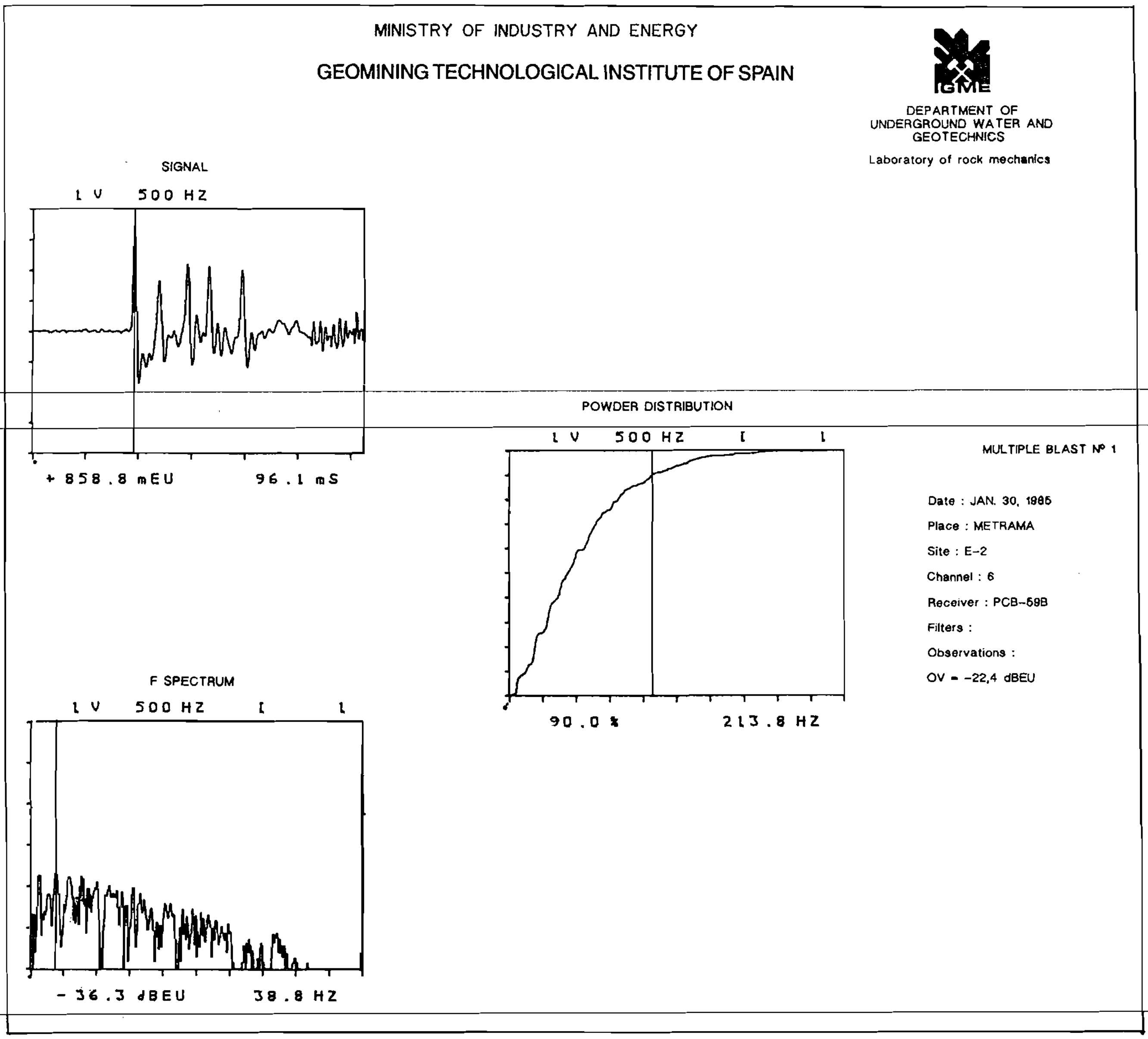
and

$$a = \text{Exponential}\left[\frac{\sum \log y}{n} - b\frac{\sum \log x}{n}\right]$$

and the lineal coefficient r from:

$$r^2 = \frac{\left[\sum(\log x) \times (\log y) - \dfrac{(\sum \log x) \times (\sum \log y)}{n}\right]^2}{\left[\sum(\log x)^2 - \dfrac{(\sum \log x)^2}{n}\right] \times \left[\sum(\log y)^2 - \dfrac{(\sum \log y)^2}{n}\right]}$$

If the Standard Deviation is also calculated, it would be possible to draw the parallel lines between which a determined number of values can be found (e.g. 95%), thus adopting a Safety Factor for the law of propagation.



MINISTRY OF INDUSTRY AND ENERGY
GEOMINING TECHNOLOGICAL INSTITUTE OF SPAIN

DEPARTMENT OF UNDERGROUND WATER AND GEOTECHNICS
Laboratory of rock mechanics

MULTIPLE BLAST Nº 1

Date : JAN. 30, 1985
Place : METRAMA
Site : E-2
Channel : 6
Receiver : PCB-59B
Filters :
Observations :
OV = -22,4 dBEU

Fig. 33.29. Example of a vibration recording obtained in a blasting for one component.

There are numerous equations (laws) that can be adjusted and, amongst all of them, the one which best suits the occasion should be chosen. This is now carried out with small computer programs which have been specially prepared.

Once the law has been estimated and the threshold of damages decided, the value of the scaled distance can be ascertained from the equation, enabling the preparation of the table of maximum co-operating charges for different distances.

For example, if the law obtained for a determined percentage of probability or safety level is:

$$v = 1400 \times DR^{-1.6}$$

where

$$DR = \frac{DS}{\sqrt{Q}}$$

and v is not to be over 30 mm/s, Fig. 33.30. The Table of Charges-Distances will correspond to $DR = 11.04$ m/kg$^{1/2}$ and therefore, the following values will exist, Table 33.4.

The method to be applied for air blast is very similar.

On the other hand, it is interesting to study how the duration of seismic excitement increases and frequency diminishes in relationship with distance to the point of blast, adjusting laws of the following type:

$$f(\text{Hz}) = K_1 \times DS^{-K_2}, \text{ and}$$

$$T_v\,(\text{s}) = K_1' \times DS^{+K_2}$$

Table 33.4.

Distance to the blast (m)	Max. co-operating charge (kg)
100	82
300	738
500	2050
700	4017

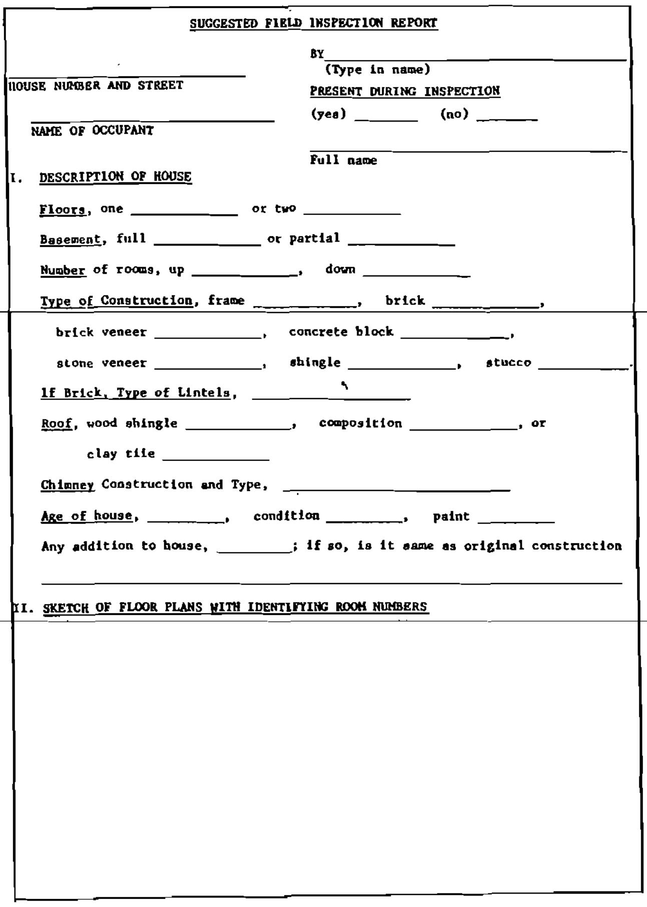
The value of T_v should refer to a same type of blast, as in multiple rounds with a total time t_v, $T_v = k \times t_v$ is fulfilled, with K taking on values of 3, 4 or even more at several hundred meters.

33.7.2 *Inspections previous to blasting*

The objective of these inspections is to compile data in a written document which gives the condition of a structure before commencing excavation work with explosives. Many buildings have cracks in unknown places and their occupants accuse the vibrations and air blast generated by blasting as the cause.

In some countries, these inspections are normal practice and, in the US, the Office of Surface Mining contemplates that any inhabitant having property at less than 800 meters from a blasting area can ask the Administration for a previous study.

The first advantage of this documentation is that it makes the residents of areas close to the blast aware of the fact that many cracks and imperfections in their buildings are originated by other than seismic causes, such as changes in weather, humidity, wind, ground conditions and the constructive quality itself. The second advantage



SUGGESTED FIELD INSPECTION REPORT

BY ________
(Type in name)

____
HOUSE NUMBER AND STREET

PRESENT DURING INSPECTION
(yes) ________ (no) ________

____
NAME OF OCCUPANT

____
Full name

I. DESCRIPTION OF HOUSE

Floors, one ________ or two ________

Basement, full ________ or partial ________

Number of rooms, up ________, down ________

Type of Construction, frame ________, brick ________,

brick veneer ________, concrete block ________,

stone veneer ________, shingle ________, stucco ________.

If Brick, Type of Lintels, ________

Roof, wood shingle ________, composition ________, or

clay tile ________

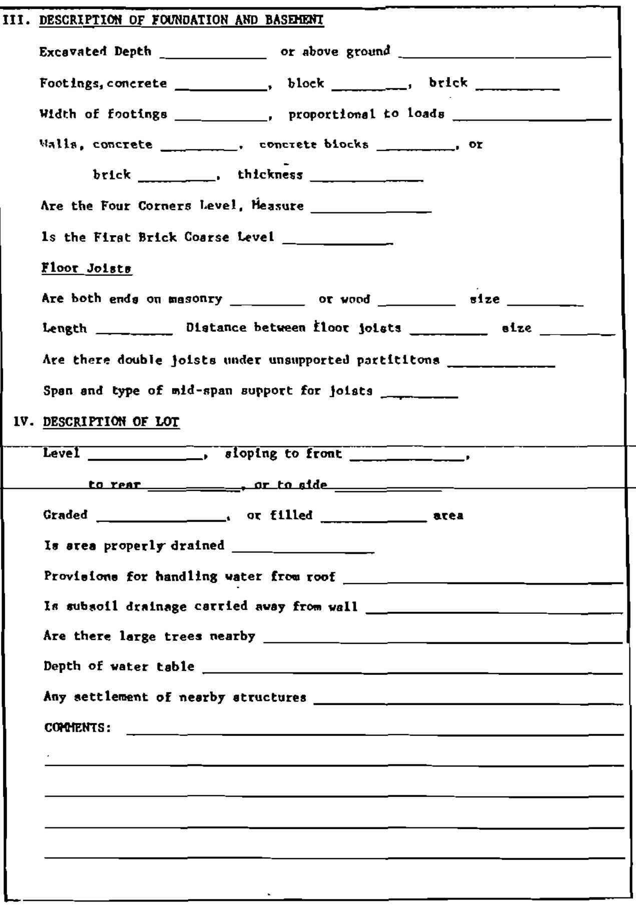
Chimney Construction and Type, ________

Age of house, ________, condition ________, paint ________

Any addition to house, ________; if so, is it same as original construction

____

II. SKETCH OF FLOOR PLANS WITH IDENTIFYING ROOM NUMBERS



III. DESCRIPTION OF FOUNDATION AND BASEMENT

Excavated Depth ________ or above ground ________

Footings, concrete ________, block ________, brick ________

Width of footings ________, proportional to loads ________

Walls, concrete ________, concrete blocks ________, or

brick ________, thickness ________

Are the Four Corners Level, Measure ________

Is the First Brick Coarse Level ________

Floor Joists

Are both ends on masonry ________ or wood ________ size ________

Length ________ Distance between floor joists ________ size ________

Are there double joists under unsupported partititons ________

Span and type of mid-span support for joists ________

IV. DESCRIPTION OF LOT

Level ________, sloping to front ________,

to rear ________, or to side ________

Graded ________, or filled ________ area

Is area properly drained ________

Provisions for handling water from roof ________

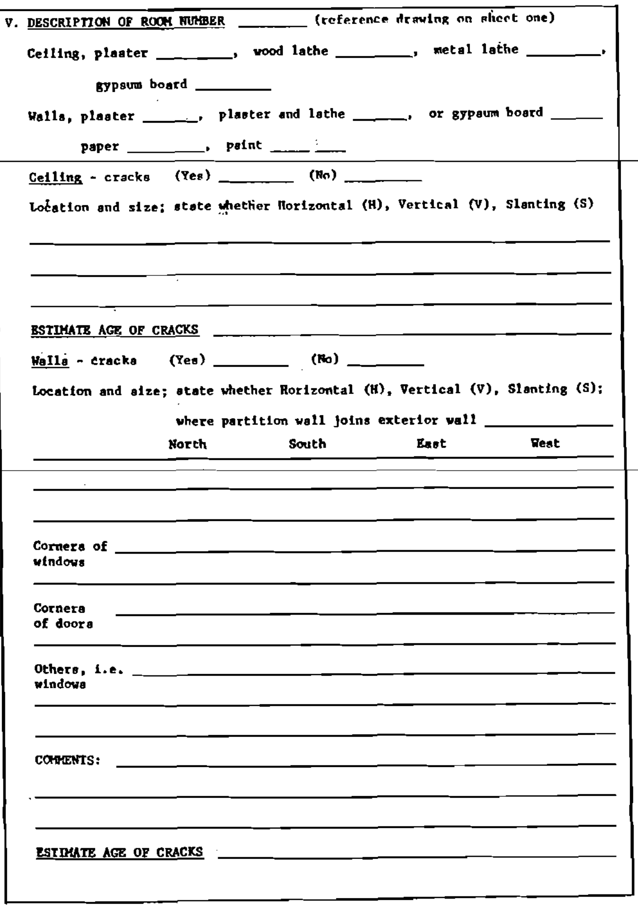
Is subsoil drainage carried away from wall ________

Are there large trees nearby ________

Depth of water table ________

Any settlement of nearby structures ________

COMMENTS: ________



V. DESCRIPTION OF ROOM NUMBER ________ (reference drawing on sheet one)

Ceiling, plaster ________, wood lathe ________, metal lathe ________,

gypsum board ________

Walls, plaster ________, plaster and lathe ________, or gypsum board ________

paper ________, paint ________

Ceiling - cracks (Yes) ________ (No) ________

Location and size; state whether Horizontal (H), Vertical (V), Slanting (S)

ESTIMATE AGE OF CRACKS ________

Walls - cracks (Yes) ________ (No) ________

Location and size; state whether Horizontal (H), Vertical (V), Slanting (S);

where partition wall joins exterior wall ________

North South East West

Corners of windows ________

Corners of doors ________

Others, i.e. windows ________

COMMENTS: ________

ESTIMATE AGE OF CRACKS ________

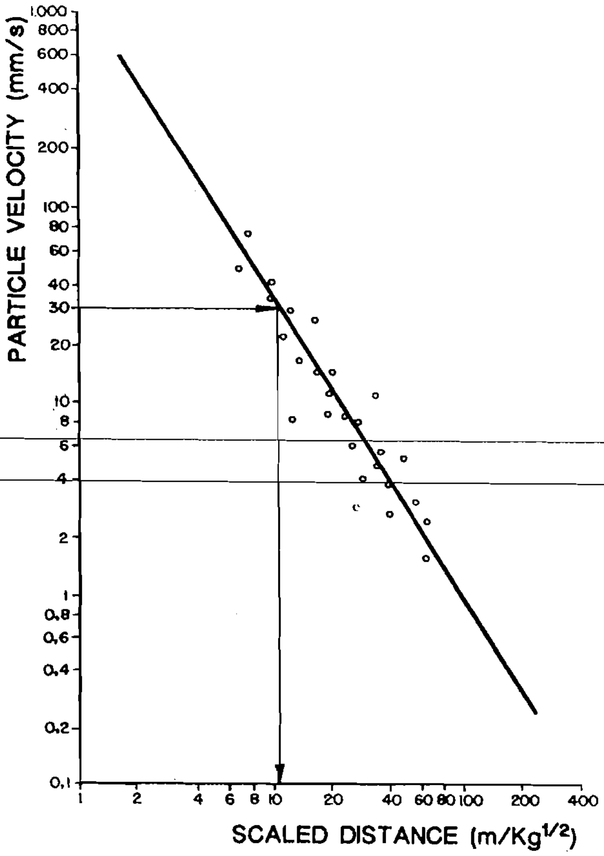


Fig. 33.30. Adjusted law of propagation.

is that the documentation can be used, if the occasion arises, to verify or contest the damage claims attributed to vibrations.

On many occasions, the initial cost of drawing up these documents is greatly compensated by the lower number of claims and conflictive situations with lawsuits between the companies and residents who are closeby. A trained person can inspect 7 or 8 homes in one day.

The procedure used for describing the condition of a structure should be as systematic and detailed as possible, writing down all visible defects and even taking photographs, if necessary. Each document should contemplate, first of all, the identity of the owners, address and situation of the residence, and the date of inspection, Fig. 33.31 shows the system used by Vibra-Tech for a study of the inside of a home. Other aspects to take into account are those which refer to the outside of the structure, garages, foundations, etc.

33.8 DAMAGE PREVENTION CRITERIA FOR BUILDINGS

33.8.1 *Building response*

Damages that appear in structures from vibration type effects depend upon the dynamic response of the building which itself, at the same time, is conditioned by various factors such as:

- Type and characteristics of the vibrations, duration, frequency, transmitted energy, etc.
- Type of ground on which the structure sits.
- Vibratory characteristics of the structural and non-structural entity of the building and modifying factors.

A parameter which is important in controlling potential damage by blast induced vibrations is their dominating frequency. In the cases where the natural frequency of the buildings is very close or equal to the dominating frequencies, a resonance phenomenon is produced with magnifying effects, Fig. 33.32.

The natural frequencies of the buildings or structures can generally be calculated analytically with simple equations, widely used in seismic engineering, such as the following:

- Buildings with prefabricated or reinforced concrete walls:

$$T_s = 0.06 \times \frac{h_v}{L_p} \times \frac{H_v}{2L_p + H_v}$$

- Buildings with framework structure of reinforced concrete:

$$T_s = 0.09 \times \frac{H_v}{L_p}$$

- Buildings with metal structure:

$$T_s = 0.10 \times \frac{H_v}{L_p}$$

In all the previous equations: t_S = Period (s), H_v = Height of building (m), L_p = Floor dimension, taken in the direction of the vibration whose effect is desired to be indicated (m), h_v = Height of each floor (m).

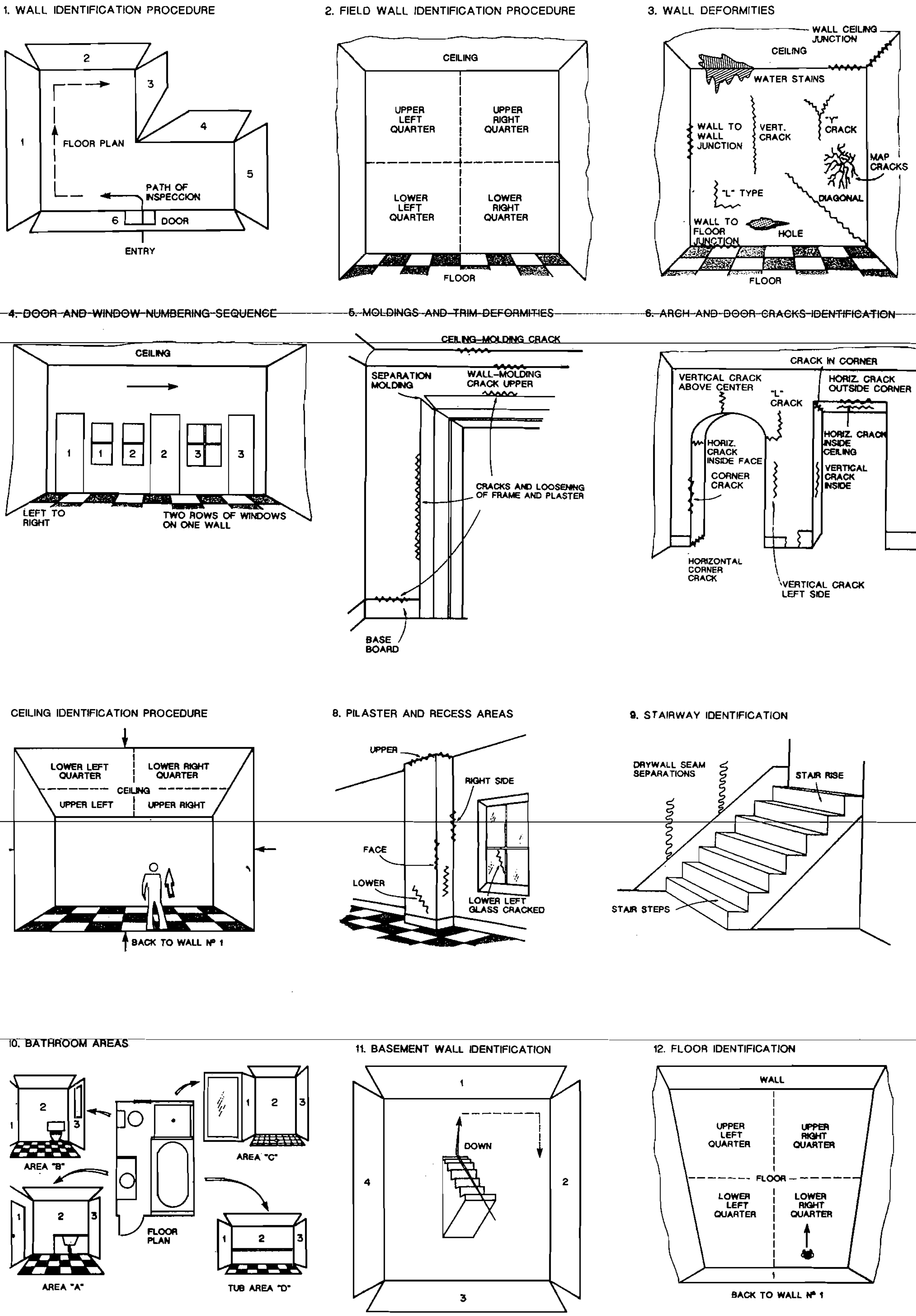
The typical frequency values are found between 5 and 15 Hz, being lower as the number of floors increase.

The ceilings and walls vibrate independently from the superstructure and usually have natural frequencies between 12 and 20 Hz.

Another parameter that is as important as the natural frequency is absorption. The common values of these coefficients in residential type structures (Dowding et al. 1980) vary around 5%.

The vibrations in buildings can be magnified due to the response of the structural elements of which they are composed, Fig. 33.33. Therefore, more attention should be paid to the times of the milisecond delay detonators. When lowering charge weights per hole and increasing blasting times, dangerous vibration frequencies may be generated if they are close to those of resonance. For example, using milisecond delay detonators of 30 ms, and leaving a number unused, a vibration of 1000/60 = 16.7 Hz is being caused, which is within the range of potential damages. This phenomenon has been proved by the authors in recorders near the blast areas. (López Jimeno and Abad, 1986).

A simple method to predict the structural response of a building to vibrations is the Fast Fourier Transform, (FFT) application. FFT informs in practice about what frequency band, and responsible wave length is needed to be omitted for avoiding damage and disturbances. FFT



1. WALL IDENTIFICATION PROCEDURE

2. FIELD WALL IDENTIFICATION PROCEDURE

3. WALL DEFORMITIES

4. DOOR AND WINDOW NUMBERING SEQUENCE

5. MOLDINGS AND TRIM DEFORMITIES

6. ARCH AND DOOR CRACKS IDENTIFICATION

CEILING IDENTIFICATION PROCEDURE

8. PILASTER AND RECESS AREAS

9. STAIRWAY IDENTIFICATION

10. BATHROOM AREAS

11. BASEMENT WALL IDENTIFICATION

12. FLOOR IDENTIFICATION

Fig. 33.31. Home inspection system.

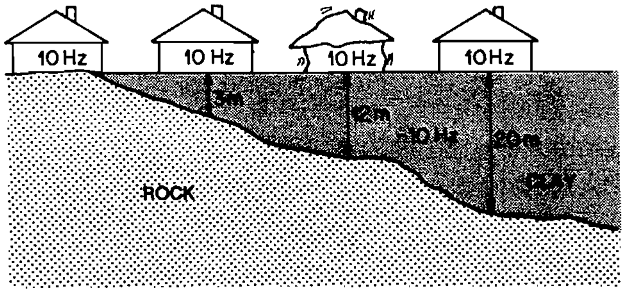


Fig. 33.32. Magnification effects when the building's natural frequency is close to the dominating frequency in the earth (Clark et al.).

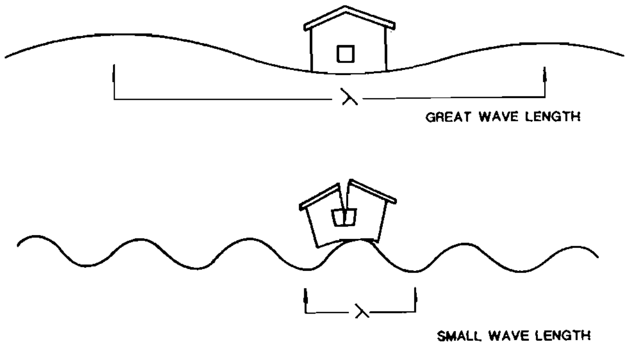


Fig. 33.35. Interaction between the building foundations and vibrating ground.

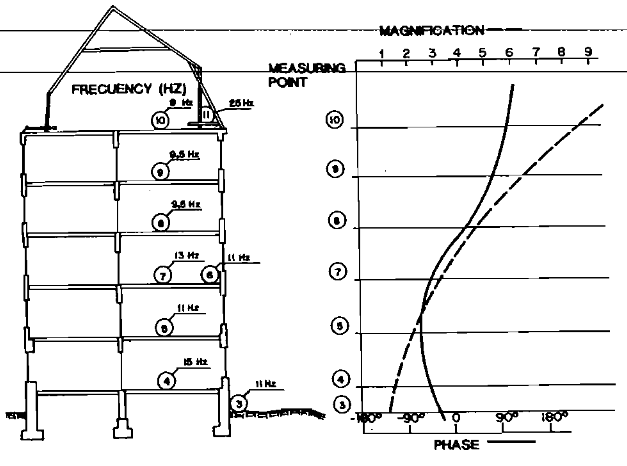


Fig. 33.33. Magnification within a building (Clark et al.).

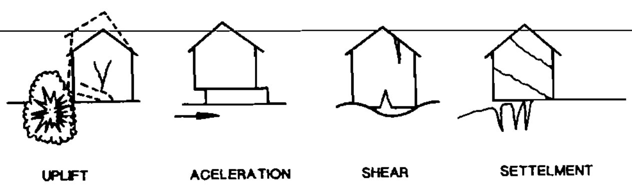


Fig. 33.36. Types of damage.

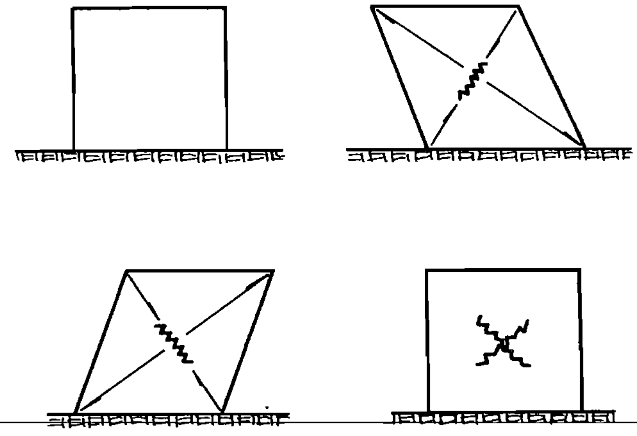


Fig. 33.37. Vibration X-crack pattern.

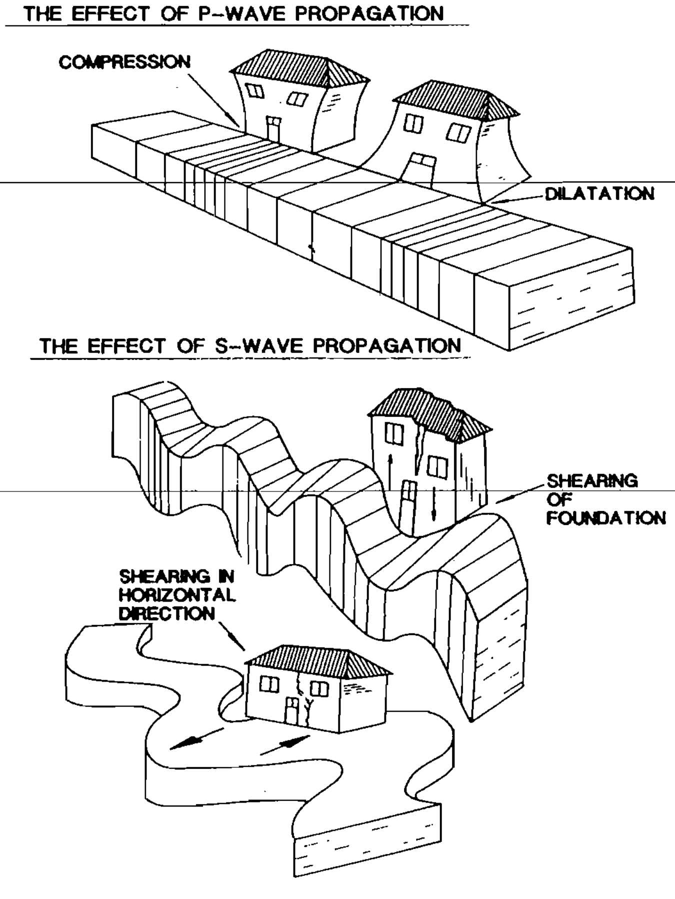

THE EFFECT OF P-WAVE PROPAGATION
COMPRESSION
DILATATION
THE EFFECT OF S-WAVE PROPAGATION
SHEARING OF FOUNDATION
SHEARING IN HORIZONTAL DIRECTION

Fig. 33.34. Effects of the *P* and *S* waves on structures.

analysis is the less costly and simplest way today and can be utilized for practically every blast design, solving successfully resonance and magnification problems.

The types of damage are diverse, Fig. 33.36: uplifts due to gas intrusion when the constructions are very close to the blast area, relative acceleration of the ground, shearings and settlements of the foundations.

When refering to the characteristic type of cracks produced by seismic motion, the most representative are those called X-cracks, because when the structures are deformed by relative movement of the bases, tensile stresses are created on the diagonals of the parallelograms which cause damage by compression, overcoming the strength of the materials, Fig. 33.37.

33.8.2 *Damage prevention criteria*

After finding the law which governs the propagation of the seismic waves in a rocky medium, the degree of maximum vibration tolerated by different types of structures near the excavation must be estimated in order to prevent damage.

The decision of which criteria or levels of vibration prevention should be adopted is usually a delicate issue. This requires expert knowledge of the mechanisms which intervene in the phenomena of blasting and the responses of structures. A risky criterium can cause damages and imperfections, whereas a conservative posture could upset or even paralize the development of mining or civil engineering activity with explosives.

The prevention criteria for vibrations produced by blasting has been subjected to study since the beginning of the century. Worthy of mention are: the investigations of Rockwell in 1927, Thoenen and Windes in 1942, who used particle acceleration as the most characteristic parameter; Crandell in 1949, who used the energy ratio, Morris in 1950, who established a new damage criterium based on the amplitude of vibration, and Langefors and Kihlström in 1958, who adopted particle velocity as the most important parameter, proposing different levels, depending upon the intensity of potential damages. Afterwards, in 1963, these authors took into consideration the type of ground upon which the structures had their foundations, proposing criteria with wider outlooks. During the decade of the sixties and seventies, numerous investigators such as Northwood, Crawford, Edwards, Duvall, Fogelson, Nicholls, etc., exposed different safety limits, all based on particle velocity, already forseeing the necessity of adjusting those prevention levels to the different types of constructions, as done by Ashley in 1976, Chae in 1978, Wiss in 1981, etc.

In another step towards developing and perfecting the criteria, apart from the type of rock under the foundations, the type of structure to be protected was introduced as another variable as important as vibration frequency, publishing the French Regulation AFTES (1976), the Standars Association of Australia Regulation, the DIN, (1983), etc. All mentioned criteria is summed up graphically in Fig. 33.38.

Afterwards, several investigators such as Dowding (1977), Medearis (1977), Maik (1979), Walker, Young and Davey (1981), Sisking, Stagg, Kopp and Dowding (1981), etc. directed their efforts towards the correlation of structure response with damages produced by different vibration intensities, through analysis of the seismic spectrums. One fact that has become more noticeable day by day in these investigations is the increasing importance of the low frequencies.

However, even though the criteria and application of techniques known in seismic engineering have evolved, the discrepancies between engineers and organisms are still quite noticeable, especially when the studies are of a local nature. It must also be noted that rarely are clear and concise recommendations or calculations given by operators who do not have a profound knowledge of the phenomenology of vibrations.

Another aspect worthy of mention is that in the majority of cases, the damage threshold is adopted for structures and buildings, without taking into account their contents. Sometimes there might be computers, electric relays or other sensitive equipment which must be protected from even lower vibration levels than those for the building itself.

Lastly, the O.S.M. (Office of Surface Mining) in the United States, in 1983, acknowledging the dependence that exists between the dominating vibration frequency and the distances to the blast area, published the following recommendations for protecting buildings near the mines, Table 33.5.

The criteria shown are not only useful as damage thresholds, but also as a starting point when recording equipment is not available. Thus, for example, when there is a house at 1000 m distance from the blast, the maximum co-operating charge recommended is:

$$DR = \frac{DS}{\sqrt{Q}} = 24.5\ \mathrm{m/kg^{1/2}};$$

$$Q = \left[\frac{DS}{DR}\right]^2 = \left[\frac{1000}{24.6}\right]^2 = 1666\ \mathrm{kg}$$

33.8.3 *Damage prevention criteria for air blast*

Air blast usually produces fewer problems than ground vibrations. Window panes usually break before structural damage occurs; cracks in the plaster, for example.

The criteria proposed by Siskind and Summers (1974), to avoid window pane breakage are shown in Table 33.6.

The probability of window pane breakage for a determined overpressure can be estimated with the equation proposed by Redpath:

$$PR_c\,(\%) = 2.043 \times 10^{-7} \times A_v^{1.22} \times \Delta P^{2.78}$$

where: A_v = Area of the window pane (m²), ΔP = Overpressure (mbar).

Special attention should be paid when comparing noise levels, as the dB(L) refer to a logarithmic scale. An overpressure of 120 dB(L) is 78.6% more than one of 115 dB(L). See Table 33.7, with the values in kPa.

Table 33.5.

Distance to blast area	Max. particle velocity (mm/s)	Recommended scaled distance when instrumentation is not available ($\mathrm{m/kg^{1/2}}$)
0 to 90 m	32	22.30
90 to 1500 m	25	24.50
> 1500 m	19	29.00

Table 33.6.

	Limit noise level		
	Lineal peak* dB(L)	C-peak dB(C)	A-peak dB(A)
Safety level	128	120	95
Precaution level	128-136	120-130	95-115
Limit level	1336	130	115

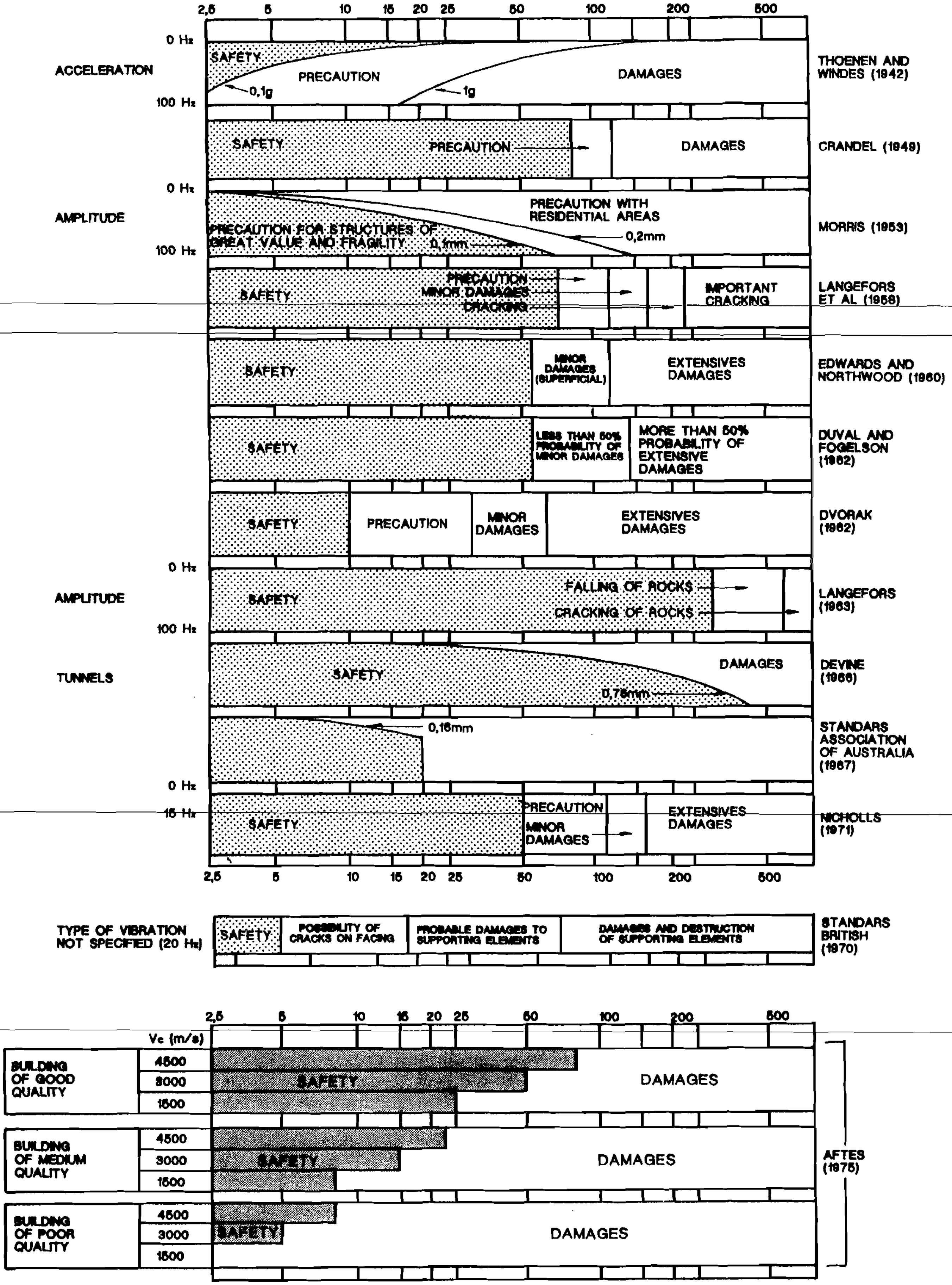
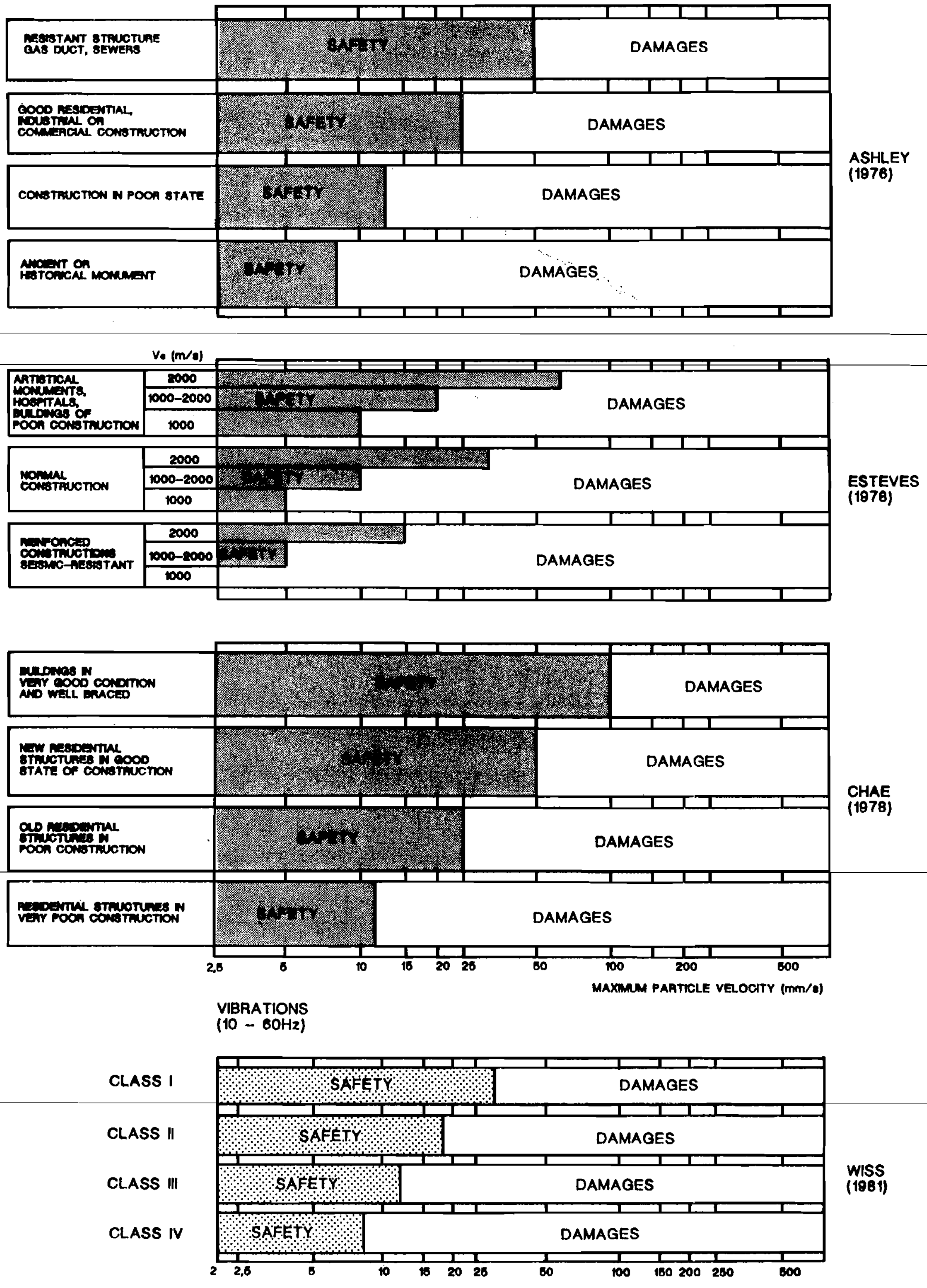


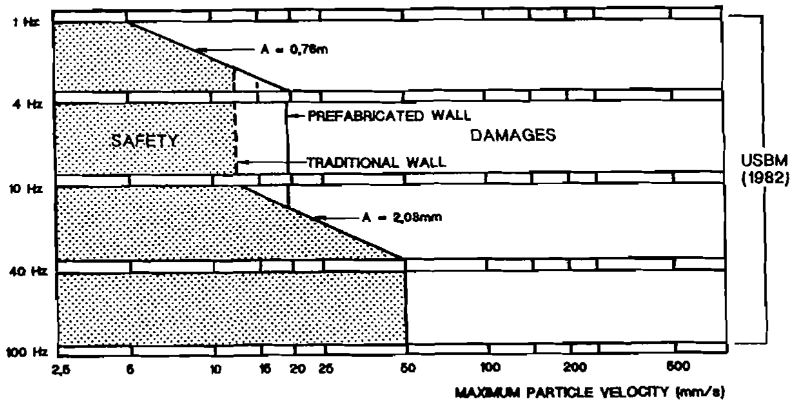
Fig. 33.38. Damage criteria.

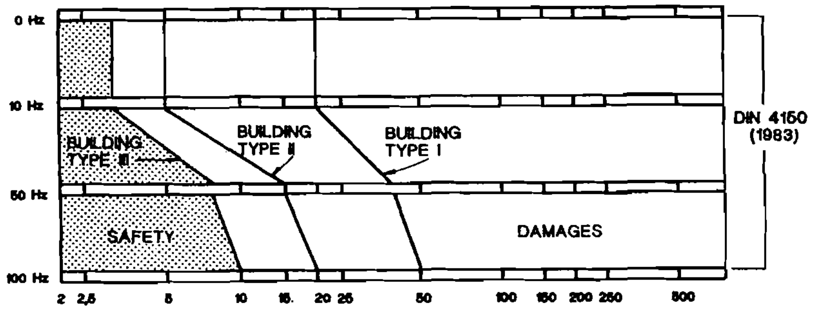


LEGEND

CLASS I : METALLIC BUILDING OR OF REINFORCED CONCRETE
CLASS II : BUILDINGS WITH OUTSIDE WALLS AND PILLARS OF CONCRETE, INSIDE WALLS OF CONCRETE OR MASONRY
CLASS III : BUILDINGS AS BEFORE BUT OF WOODEN STRUCTURE AND WALLS OF MASONRY
CLASS IV : CONSTRUCTION VERY SENSITIVE TO VIBRATIONS; OBJECTS OF HISTORICAL INTEREST

Fig. 33.38. Damage criteria (cont.).





REGULATION DIN 4150 (RESULTING v)

TYPE I : PUBLIC OR INDUSTRIAL BUILDING
TYPE II : APARTMENT BUILDING OR SIMILAR BUILDINGS WITH STUCCO OR PLASTER
TYPE III : HISTORICAL-ARTISTICAL BUILDINGS OR DUE TO THEIR CONSTRUCTION ARE SENSITIVE TO VIBRATIONS AND DO NOT PERTAIN TO GROUPS I AND II

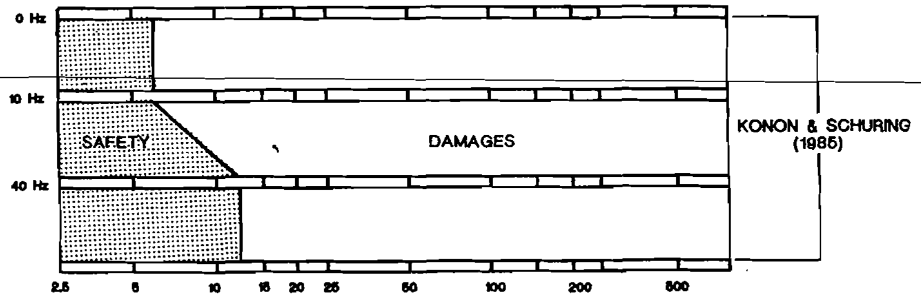


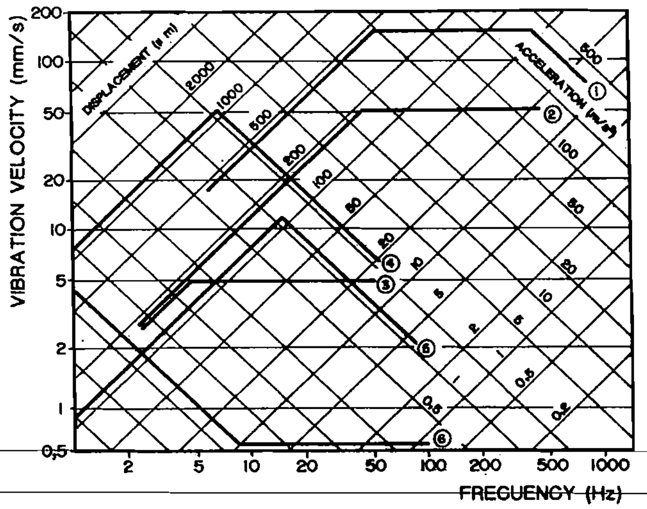
Fig. 33.38. Damage criteria (cont.)

Table 33.7.

Overpressure	Probable effect	
180 dB(L)	20.0 kPa	Important damage to conventional structures
> 170	> 6.3	Cracks will appear in plaster
170	6.3	Many window panes will break
150	0.63	Some window panes will break
140	0.2	Probable breakage of large window panes
136	0.13	Limit of air blast proposed by U.S.B.M.
120	0.02	Complaints
115	0.0112	< 6% of overpressure which can cause breakage of large window panes

Table 33.8 (Baker, 1973).

Description	Intensification factor
Simple negative gradient	0
Simple positive gradient	5
Zero gradient near the surface and with positive gradient above	10
Negative gradient near surface with strong positive gradient above	100



1. DIRECT DAMAGE ON BUILDINGS FROM BLAST GENERATED VIBRATIONS
2. UPPER LIMIT RECOMMENDED FOR BLAST
3. UPPER LIMIT RECOMMENDED FOR SINKING PILES VIBRATORY COMPACTORS, DEEP DYNAMIC COMPACTORS AND TRAFFIC ON WHEELS
4. MAXIMUM VALUE FOR IBM COMPUTERS IF THE DURACTION OF THE VIBRATION IS UNDER 5s
5. MAXIMUM VALUE FOR IBM COMPUTERS IF THE DURACTION OF THE VIBRATION IS OVER 5s
6. LIMIT OF HUMAN PERCEPTION

Fig. 33.39. Damage criteria.

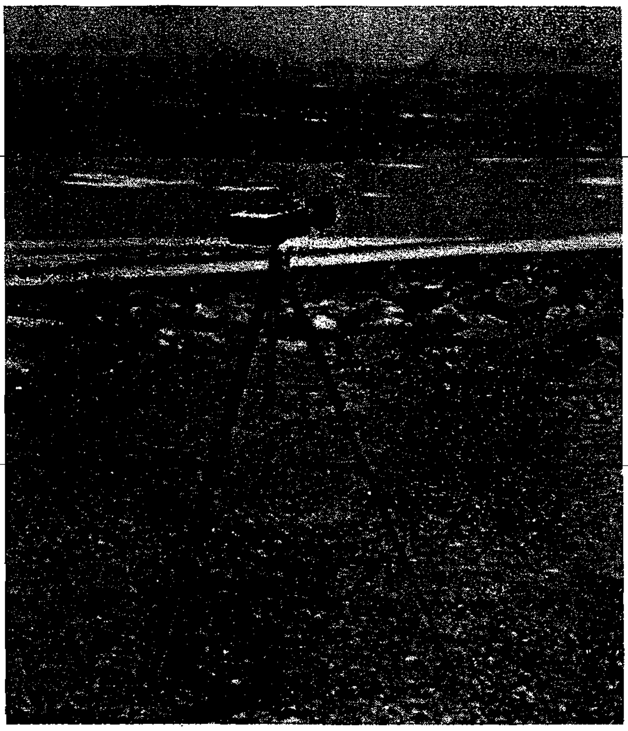


Photo 33.5. Sonometer installed to measure air blast.

Other important aspects to take into account are the atmospheric conditions at the moment of the blast. In Table 33.8, five different situations are shown and the intensification factors of air blast that can be expected.

33.9 EFFECTS OF VIBRATIONS AND AIR BLAST ON PEOPLE

One of the factors to be considered when blasting is the physiological response of human beings, as with levels under the maximum admissible for prevention of damage to structures, there can be an index of perception which could make people think of probable damage, Fig. 33.40. Therefore, it is frequent that in many projects the vibration thresholds are based more on human response than on the probability of damages.

There are numerous regulations on human response to vibrations, the two most important being ISO-2631 and DIN-4150. Other investigations such as those of Reiher-Meister, Crandell, Goldman, Rathbone, etc., who represent graphically where different levels of perception are established in function with vibration intensity and frequency, Fig. 33.41.

An analytical procedure of estimation is proposed by Steffens (1974), based on the calculation of a parameter *K*.

$$K = \frac{0.005\,A \times f^2}{(100 + f^2)^{1/2}} = \frac{0.8\,v \times f}{(100 + f^2)^{1/2}} = \frac{0.125 \times a}{(100 + f^2)^{1/2}}$$

where: f = Frequency (Hz), A = Amplitude (μm), v = Particle velocity (mm/s), a = Acceleration (mm/s^2).

According to this value of *K*, the levels of perception of Table 33.9, are distinguished

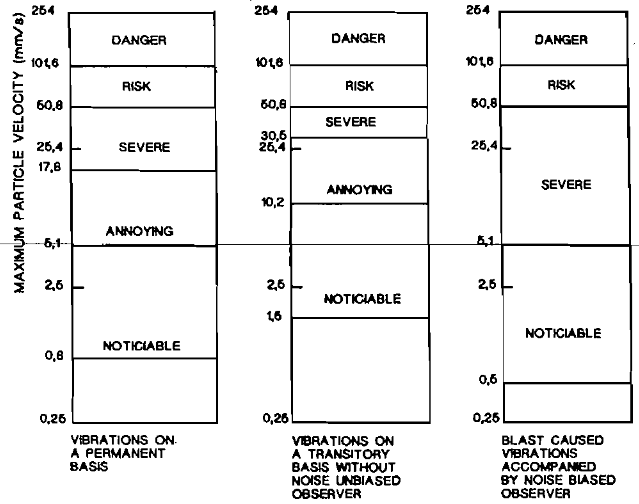


Fig. 33.40. Human response to vibrations, according to whether they are accompanied by noise or not (Oriard).

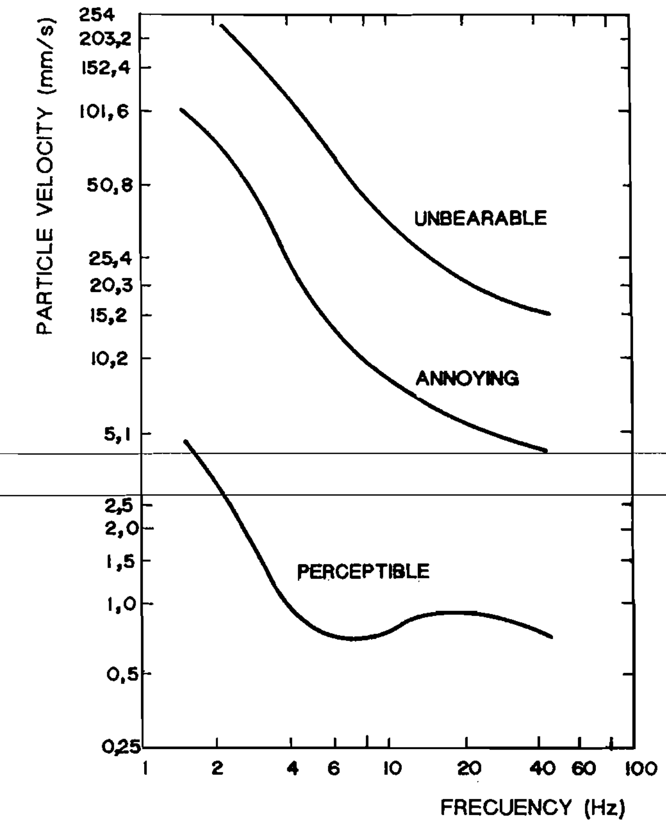


Fig. 33.41. Human response to vibration according to Goldman (1948).

Table 33.9.

Value of K	Level of perception
< 0.1	Not perceptible
0.1	Almost perceptible
0.25	Barely perceptible
0.63	Perceptible
1.6	Easily perceptible
4.0	Highly detectable
10.0	Severely detectable

33.10 EFFECTS OF VIBRATIONS ON ROCKMASSES

Vibrations have two fields of action on rock masses. On one hand they affect the integrity of the rocks or their compressive strength parameters and, on the other, can provoke wall or slope collapse when unstabilizing actions are introduced.

In the first instance, the critical vibration velocity can be determined after finding the longitudinal wave propagation velocity in the rock mass, the density and the tensile strength of the rock.

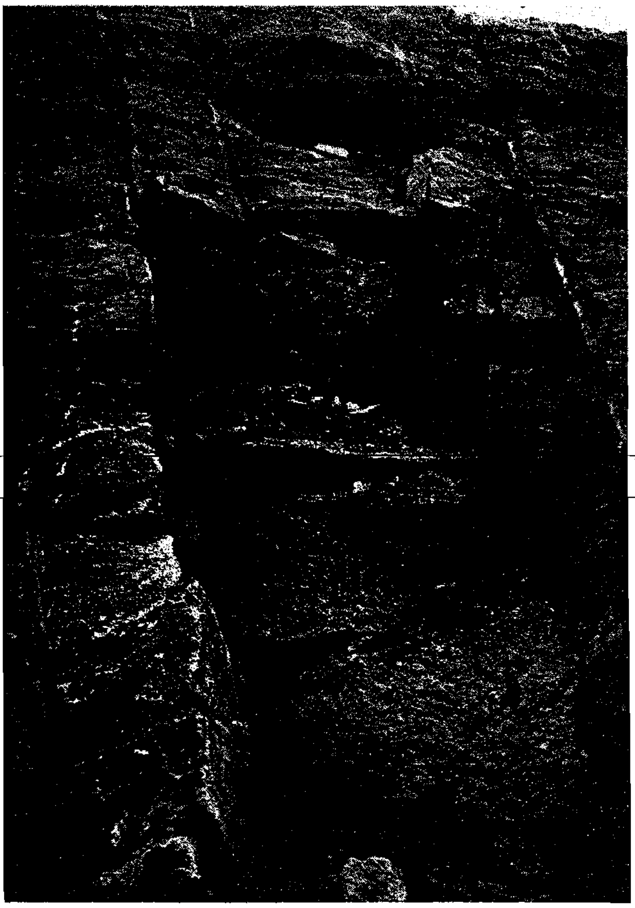


Photo 33.6. Damage produced in a presplit blast caused by overcharging.

$$RT = \rho_r \times v_{crit} \times VC$$

$$v_{crit} = \frac{RT}{\rho_r \times VC}$$

where: RT = Tensile strength, ρ_r = Density of the medium, VC = Propagation velocity of longitudinal waves.

Thus, for a rock with $\rho_r = 2.6$ t/m^3 and $VC = 4500$ m/s, the following exists:

$$v_{crit}\,(\text{mm/s}) = \frac{RT\,(\text{MPa})}{0.117}$$

According to Oriard (1970), the damage threshold in rock slopes is around 60 cm/s of particle velocity.

Afterwards, Bauer and Calder (1971), give the criteria shown in Table 33.10.

Fig. 33.42, gives, in a general sense, the predictable damages due to effect of vibrations in function with the maximum charge weight per unit of delay and the distance from the center of gravity of the blast to the recording point.

Fig. 33.43, shows a procedure to estimate the damages to rock masses from blast vibrations.

When referring to wall stability, this can be determined

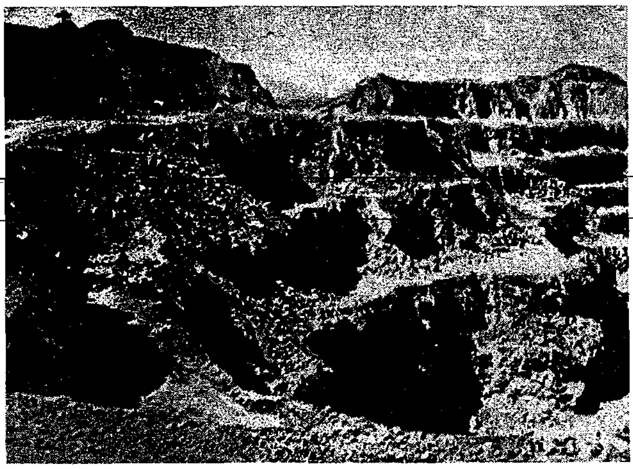


Photo 33.7. Backbreak and face loose rock on final pit slope.

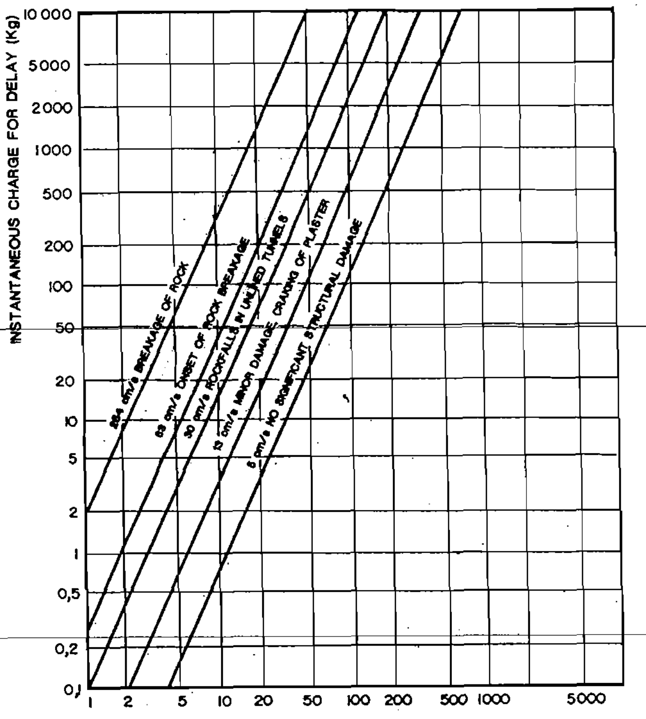


Fig. 33.42. Relationship between maximum charge weight per delay, distance and peak particle velocity.

Table 33.10.

Particle velocity (cm/s)	Predictable damages
< 25	No danger in sound rock
25-60	Possible sliding due to tensile breakage
60-250	Strong tensile and some radial cracking
> 250	Complete breakup of rock masses

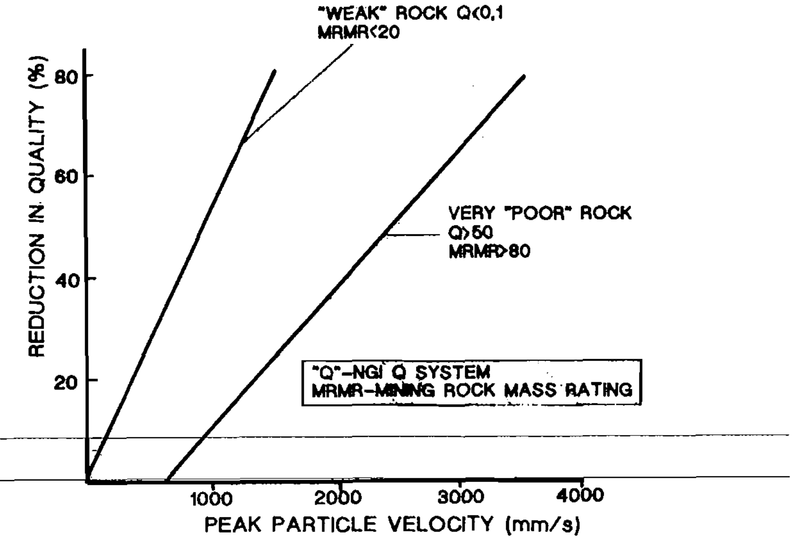


Fig. 33.43. Loss of rock mass quality according to vibration level.

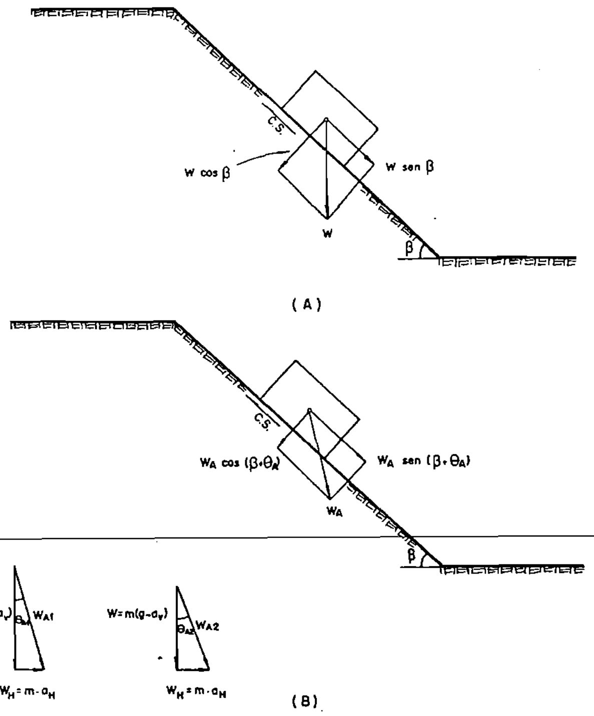


Fig. 33.44. Unstabilizing effect of vibrations in a block resting on a slope.

by the relationship between the active forces, which tend to produce sliding or failure, and the resisting forces, which oppose the movilization of the masses implicated. Although the behavior of a wall when confronted with dynamic effects is complex, owing to the numerous factors that concur, one of the simplified methods to calculate the safety coefficient consists in supposing that the acceleration or velocity due to the seism of the blast is changed into a static force in a determined direction and is proportional to the weight of the sliding mass.

In the case of a block resting on an inclined plane, Fig. 33.44, the equation that gives the Safety Factor, *SF* depreciating the effect of the vertical component of vibratory movement, is:

$$FS = \frac{C_h \times S_p + W_A \times \cos(\beta + \theta_A) \times \mathrm{tg}\,\phi}{W_A \times \mathrm{sen}\,(\beta + \theta_A)}$$

where: C_h = Cohesion, S_p = Contact area of the block, W_A = Weight of the block, β = Slope angle, ϕ = Friction angle, θ_A = Angle caused by longitudinal component of vibrations.

In the particular case of zero cohesion and with the following values: $\beta = 32°$ and $\phi = 37°$, the Safety Factor is 1.2, but if the vibrations act with a longitudinal component v_H = 6 mm/s, with a frequency of 25 Hz, *SF* changes to be 0.98 and block sliding is produced.

Depending upon the type of failure, calculation models can be developed to determine Safety Factors for different levels of vibrations or viceversa, Fig. 33.45.

33.11 EFFECT OF VIBRATIONS ON FRESHLY POURED CONCRETE

In actual practice, numerous occasions arise when it is necessary to build concrete structures at the same time when excavations by blasting are being carried out. For example, linings during tunnel driving, foundations for the primary crushing buildings near open pits, etc.

Fig. 33.46, shows prevention criteria given by Oriard depending upon curing or hardening time of the concretes, although such recommendations cannot be made extensive to all types of concrete.

As can be observed, during the hardening period of 0 to 4 hours, the concrete is still not hard and the admissible levels are relatively high. From 4 to 24 hours, it begins to harden slowly, and after 7 days it reaches a strength that is approximately ⅔ of the final product (28 days), allowing a progressive intensification of the vibrations.

The empirical equations which can be used for an orientative calculation of the maximum co-operative charges, according to age of concrete and distances to blast are:

Fill and mass concrete

$$Q = 38.20 \times 10^{-3} \times DS^{1.86} \times K$$

$(DS$ in m and Q in kg$)$

where: K = 1.0 for t = 0-4 hours, K = 0.16 for t = 4-24 hours, K = 0.3 for t = 1-3 days, K = 0.7 for t = 3-7 days, K = 2.3 for t = 7-10 days, K = 5.5 for t = + 10 days.

Reinforced or structural concrete

$$Q = 14.55 \times 10^{-3} \times DS^{1.86} \times K$$

$(DS$ in m and Q in kg$)$

where: K = 1.0 for t = 0-4 hours, K = 0.08 for t = 4-24 hours, K = 0.37 for t = 1-3 days, K = 1.0 for t = 3-7 days, K = 3.0 for t = 7-10 days, K = 7.58 for t = + 10 days.

Other factors to take into account are the characteristic frequencies of the vibrations, external hardening conditions, areas of rock-concrete contact, etc.

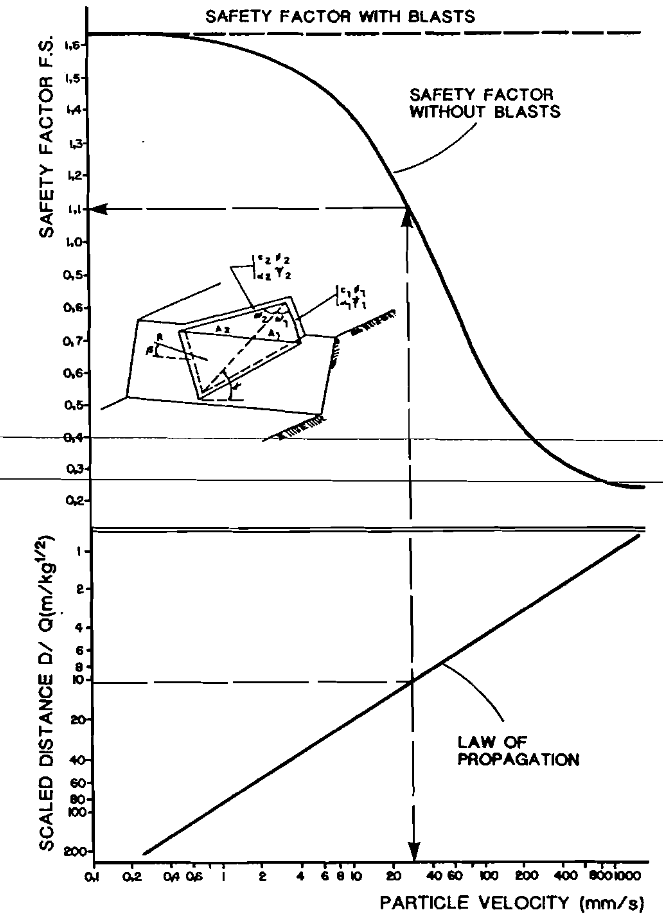


Fig. 33.45. Variation of the Safety Factor for a block with a wedge cut in function with scaled distance.

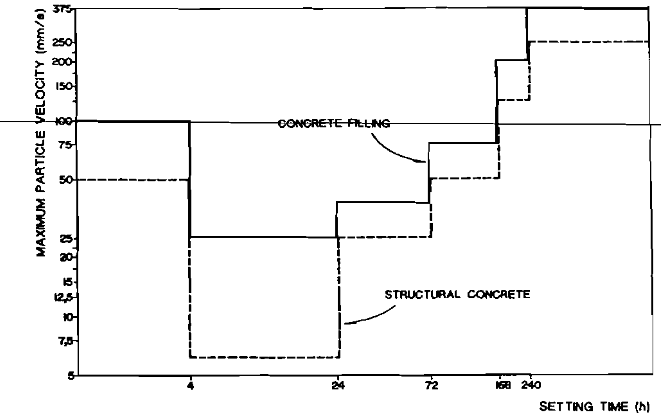


Fig. 33.46. Maximum particle velocity in function with hardening time.

On the other hand, Isaac and Bubb (1981), summed up all their experiences and those of Scandinavian investigators in a graph where, according to the strength acquired by the concrete, the maximum vibration level is determined.

In the construction of some nuclear plants in Spain the following criteria have been used:

Fill concrete

$$v_{adm}\,(\mathrm{mm/s}) = 100 \times \frac{RC(t)}{15} \leq 100$$

where: $RC(t)$ = Strength acquired by the concrete after a time t (MPa).

With the limitations:

- Time passed after pouring the concrete ≥ 8 h.
- Maximum particle velocity ≤ 100 mm/s.

Structural concrete

$$v_{adm}\,(\mathrm{mm/s}) = 60 \times \frac{RC(t)}{25} \leq 100$$

With the same limitations as before.

33.12 RECOMMENDATIONS FOR REDUCING GROUND VIBRATION AND AIR BLAST LEVELS

Although each case should be carefully analyzed, the principal measures that can be taken for reducing blast generated vibrations are:

- Minimizing the explosive charge per milisecond delay: Reducing the drilling diameter; Shortening the length of the holes; Decking the charges in the holes and initiating them at different times; Utilizing the maximum number of detonators or delay times possible, with sequenced explosives or milisecond delays if the commercial series of electric blasting caps is surpassed, Figs 33.48 and 33.49.

- Reduce the number of blastholes having instantaneous detonators, as these give higher dispersion that the highest numbers of the series.

- Choose an effective delay time between holes and rows which avoid wave interaction and give good rock displacement.

- Set the initiation sequence in a way that it progresses away from the structure to be protected, Fig. 33.50.

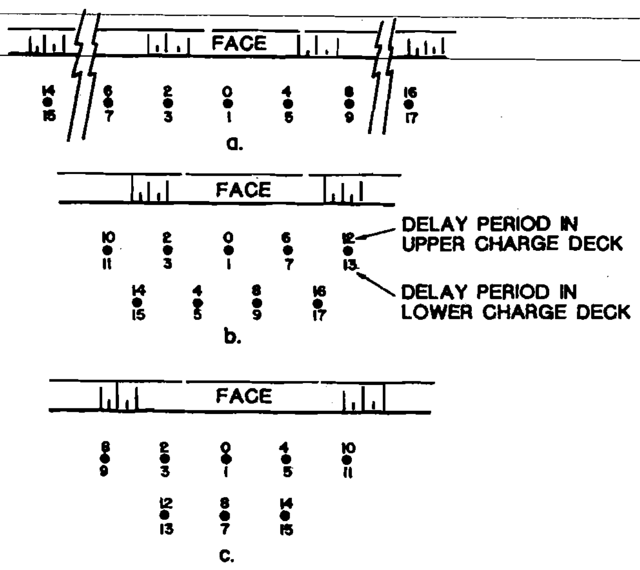

FACE
a.
FACE
DELAY PERIOD IN UPPER CHARGE DECK
DELAY PERIOD IN LOWER CHARGE DECK
b.
FACE
c.

Fig. 33.48. Blasts with decked charges in the holes.

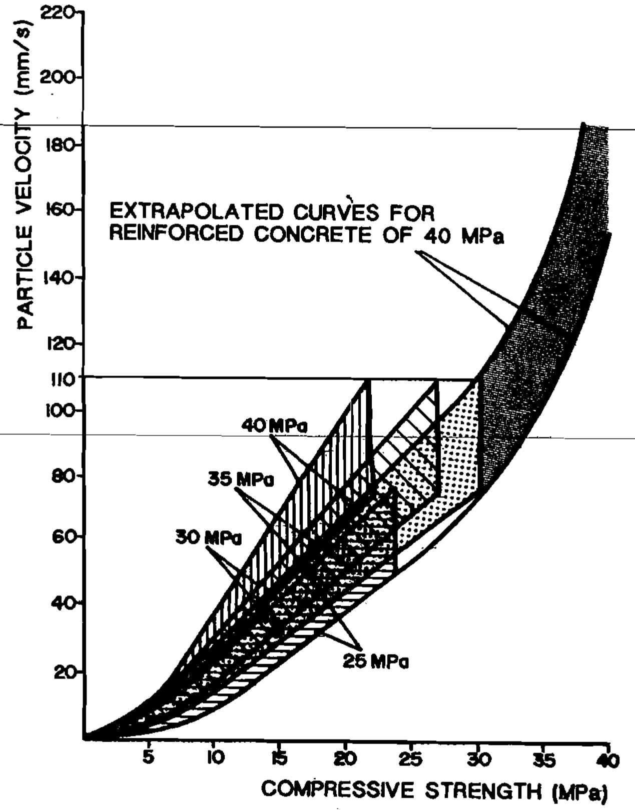
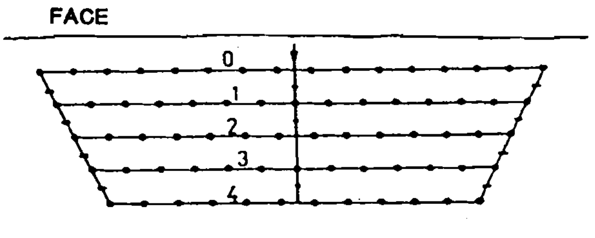
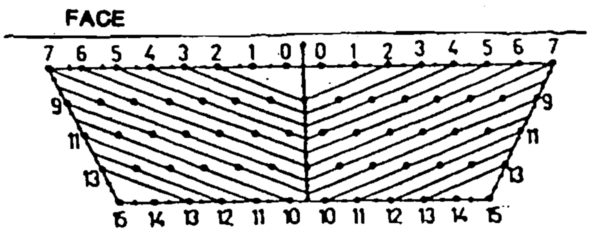


Fig. 33.47. Admissible vibration levels depending upon the strength of the concrete.



a. STAGGERED IN LINE



b. STAGGERED "V2"

Fig. 33.49. Multiple blasts with the same number of holes and different durations.

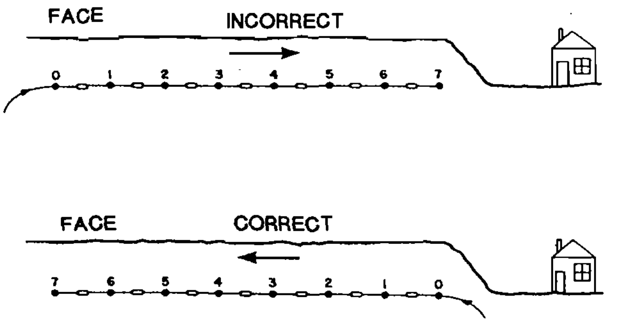


Fig. 33.50. Initiation sequence in relation to the structure to be protected.

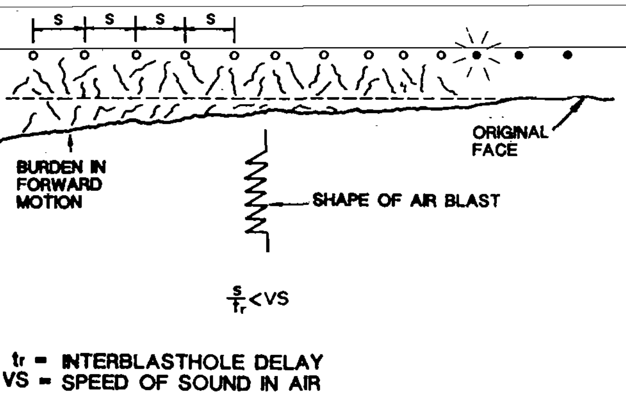


Fig. 33.51. Blast progression along a face and simulation of air blast.

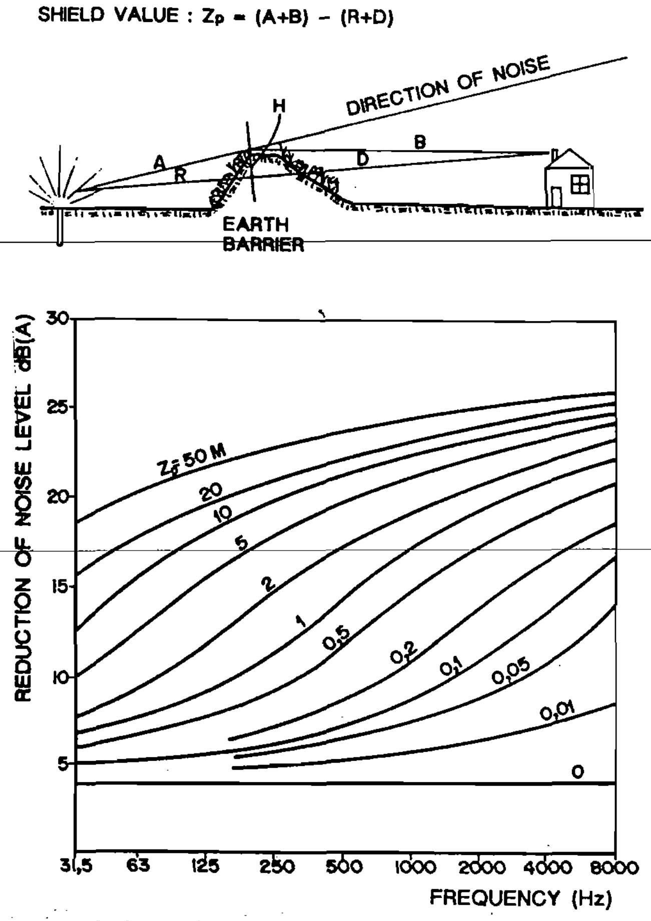


Fig. 33.52. Interposing shields between the blasts and receiving points.

– Use the adequate powder factor, as when it is lowered the charge confinement can increase and, consequently, so will the intensity of vibrations. Obviously, an excessive consumption will create an unnecessary overload, accompanied by great disturbing effects.

– Place the pattern with a relationship $H/B > 2$.

– Control drilling so that the patterns coincide with the nominal ones.

– Use only the subdrilling necessary to acheive good breakage.

– Use the largest possible face blast area.

– Create shields or discontinuities between the structures to be protected and the masses to be blasted.

As with ground vibrations, the recommendations for air blast reduction are:

– Minimize charge weight per milisecond delay. (See the corresponding part about ground vibrations).

– Choose delay times so that the blast progresses along the face at a velocity lower than that of sound in the air (< 340 m/s), Fig. 33.51.

– Increase confinement of the explosive charges with long stemming heights > $25D$, but not excessive, and use adequate inert material.

– Avoid using detonating cord, and when it is necessary, cover it with fine sand of a minimum thickness of 7 to 10 cm.

– Never fire blasts when the direction of the wind is critical.

– Select patterns and sequences that avoid cooperative wave interaction.

– Inspect the state of the faces before blasting in order to correct the charges with in the blastholes with burdens that are under the nominal.

– Control the explosive charge in ground with solution cavities to eliminate pocket concentrations.

– Place earth or other types of shields between blast and receiving point, Fig. 33.52.

33.12.1 *Reducing vibrations with precision detonators*

The effect of lineal interference or superposition of the wave trains generated by different sequenced explosive

Photo 33.8. Face displacement in a one-hole blast for a vibration study.

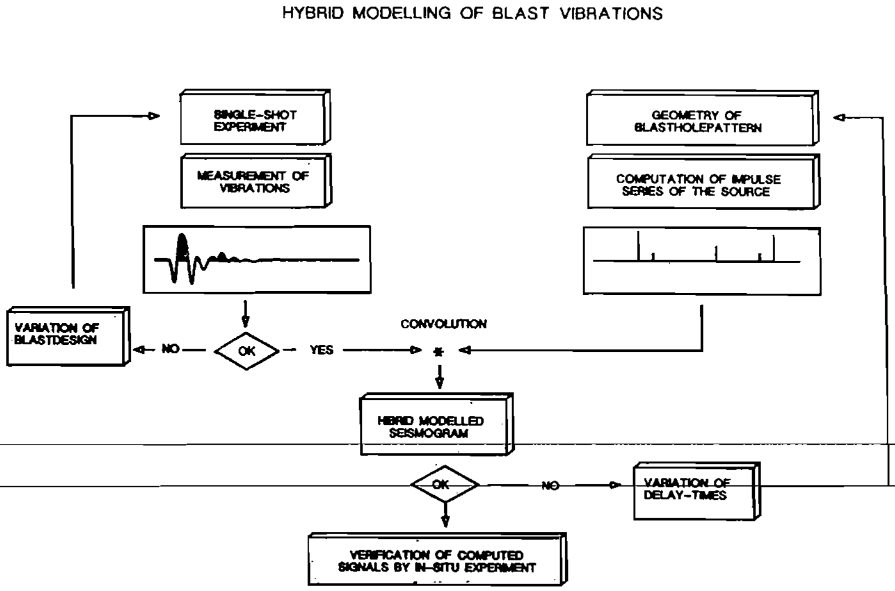

HYBRID MODELLING OF BLAST VIBRATIONS
SINGLE-SHOT EXPERIMENT
MEASUREMENT OF VIBRATIONS
GEOMETRY OF BLASTHOLEPATTERN
COMPUTATION OF IMPULSE SERIES OF THE SOURCE
VARIATION OF BLASTDESIGN
NO
OK
YES
CONVOLUTION
*
HIBRID MODELLED SEISMOGRAM
OK
NO
VARIATION OF DELAY-TIMES
VERIFICATION OF COMPUTED SIGNALS BY IN-SITU EXPERIMENT

Fig. 33.53. Phases of vibration simulation in multiple blastings.

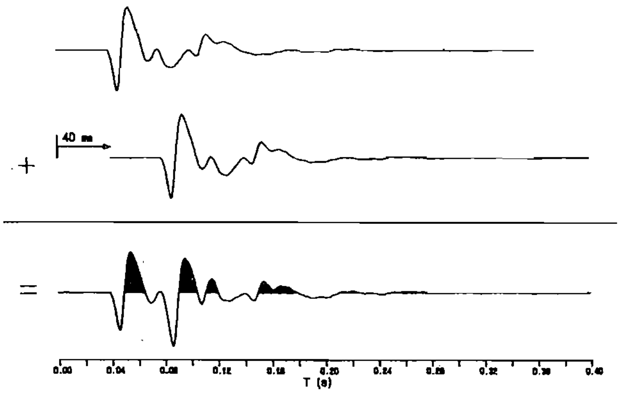


Fig. 33.54. Seismograph resulting from the superposition of two wave trains that are 40 ms apart.

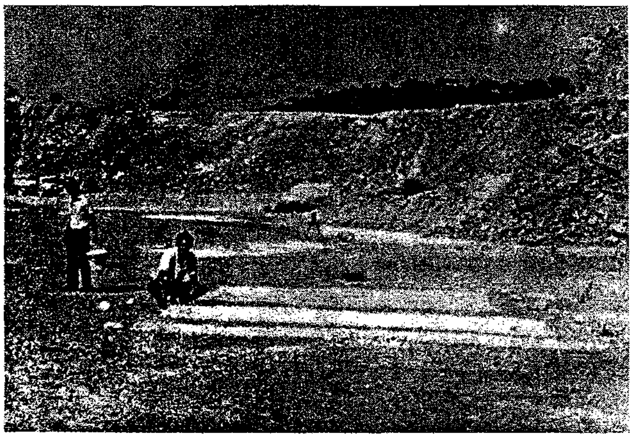


Photo 33.9. Field tests to measure the effectiveness of noise and air blast reduction in a detonating cord covered with sand.

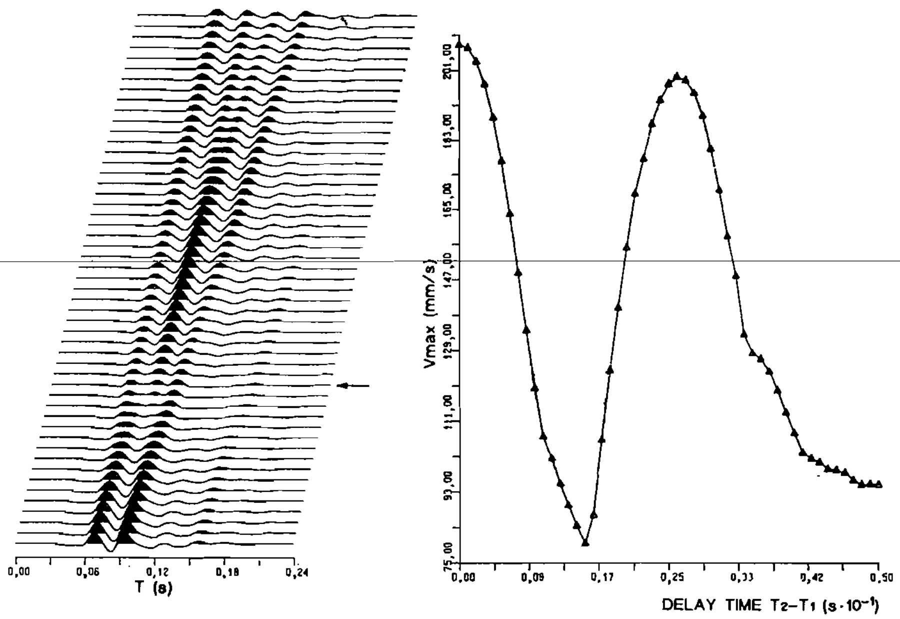


Fig. 33.55. Results of the superposition of two signals on the peak vibration value when varying delay times.

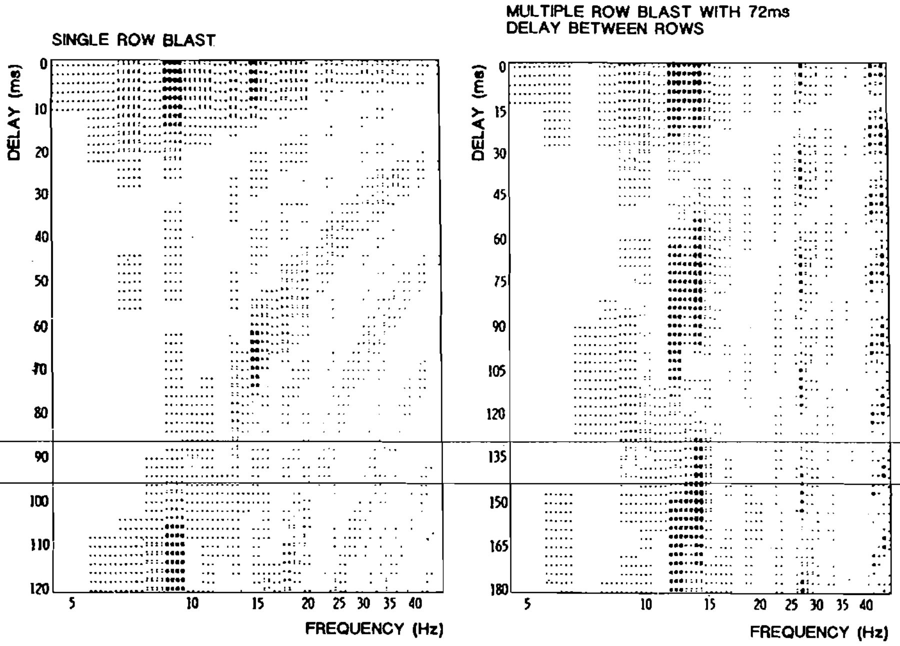


Fig. 33.56. Frequency patterns vs delay time. (a) Row with 4 blastholes with 2 ms increases between charges, (b) Two rows of 4 blastholes, with 72 ms delay between holes in row, and 3 ms increase between rows.

charges is a phenomenon which has drawn much attention lately. Supposing that each hole of a blast produces the same vibration, but delayed in time by sequenced initiation, it is possible to simulate the recording that would be obtained – with its maximum particle velocity and dominating frequencies – by combining the vibrations of a group of blastholes with a given geometry and initiation sequence.

In Fig. 33.53, a simulation procedure is given for the vibrations of a blast, having on hand the actual recording of the signal produced by only one hole.

As an example of these simulations, Fig. 33.55 shows the result of the superposition of two equal wave trains between which exists a time difference of 40 ms.

In practice, the milisecond detonators give a dispersion (cap scatter) in initiation times, increasing with the higher series numbers. For this reason, the computer simulators should be more probabilistic than deterministic, and the Monte Carlo method can be applied to establish the initiation times of each charge by creating aleatory numbers and by using the functions of density of the nominal milisecond delay times.

Recently, with the development of high precision detonators, the old idea of achieving the superposition or destructive interferences of vibrations so that that the peaks and valleys of two waves would be nullified, thus reducing vibrations, has taken on importance and constitutes a field of investigation that is reaping benefits.

The use of these electronic accessories, along with sequential blasting machines, gives an infinite number of combinations. The simulation of the results obtained simplifies making the most appropiate choice to reduce vibration levels and control frequency.

Fig. 33.55 gives the results of variation in delay timing, with increases of 1 ms, in the superposition of two signals. As can be observed, the delay of 15 ms gives the lowest maximum vibration velocity.

In the same manner, the spectral analysis of Fourier can be carried out in order to determine the dominating frequencies that would be generated. Fig. 33.56 shows two simulations that correspond to a single row blast and to another multiple blast where two different delay intervals, multiples of 2 and 3 ms respectively, are studied. Each row of the graph represents the spectra of frequencies with which the theoretical optimum sequence can be determined to avoid low frequencies, proven to be the most dangerous, in a blast of prefixed geometry.

REFERENCES

Abad, M: *Puesta a Punto de un Equipo de Registro y Análisis de Voladuras. Resultados Obtenidos en Rocas Igneas. Simpósium sobre el Uso Industrial del Subsuelo.* 1981

Allard, P. et al.: *Etude des Vibrations Engendreés par les Tirs de Mines.* Scetaroute, Février, 1986.

Anderson, D.A. et al.: *A method for site-specific prediction and control of ground vibrations from blasting.* S.E.E., 1986.

Anderson, D.A.: *The 8 Minisecond Criterion. Have we delayed too long in questioning it?* S.E.E., 1989.

Andrews, A.B.: *Design criteria for sequential blasting.* E.I. Dupont de Nemours Co., 1980.

Andrews, A.B.: *Control of ground vibration from surface blasting using computer simulations to evaluate the effects of cap scatter.* S.E.E., 1990.

Ashley, C.: *Blasting in urban areas.* Tunnels & Tunneling, Vol. 8, Sept. 1976.

Barkley, R.C. et al.: *Ground and air vibrations caused by surface blasting.* N.T.I.S., 1983.

Birch, W.J. et al.: *Predictions of ground vibrations from blasting on opencast sites.* NCB Opencast Executive, 1982.

Bollinger, G.A.: *Blast Vibration Analysis.* 1971

Clark, D. et al.: *Vibration: Its effect & measurement techniques at or near dwellings.* Technical Report, Nitro Consult. 1982.

Chae, Y.S.: *Design of excavation blasts to prevent damage.* Civil Engineering, ASCE, vol. 48, April, 1978.

Dowding, C.H.: *Blast vibration monitoring and control.* Prentice-Hall, Inc., 1985.

Edwards, A.J. & T.D. Northwood: *Experimental studies of the effects of blasting on structures.* The Engineer, vol. 210, Sept. 1960.

Esteves, J.M.: *Control of vibrations caused by blasting,* Memoria 498, Laboratorio National de Engenharia civil, Lisboa, 1978.

Fogelson, D.E.: *US Bureau of Mines Research on vibrations from blasting.* L'Industrie Minerale, 1971.

German Institute of Standards: *Vibration of building. Effects on structures.* Pastfach 1107, DIN 4150.

Ghosh, A. & J.K. Daemen: *A simple new blast vibration predictor-based on wave propogation laws.* 24th, US Symposium on Rock Mechanics, 1983.

Hagan, T.N.: *The design of blasting procedures to ensure acceptable noise air blast and ground vibrations in surface coal mining.* Environmental Control in Coal Mining, 1980.

Hendron, A.J. & L. Oriard: *Specifications for controlled blasting and civil engineering projects.* Proceedings North American Rapid Excavation and Tunneling Conference, AIME, 1972.

Hidalgo, E. et al.: *La Medida de la Onda de Baja Frequencia Producida por las Voladuras.* VIII Simposio Nacional sobre Reconocimiento de Macizos Rocosos, 1984.

Hinzen, K.G. et al.: *A new approach to predict and reduce blast vibration by modelling of seismograms and using a new electronic initiation system. S.E.E, 1987.*

Holmberg, R. & P.A. Persson: *Design of tunnel perimeter blasthole patterns to prevent rock damage.* Proc. Tunnelling '79.

Lande, G. et al.: *Controlled Tunnel Blasting.* Tunnels & Tunnelling, 1982.

La Orden, L.: *Acciones Sísmicas en la Edificación.* Instituto Eduardo Torroja, 1973.

Leet, L.D.: *Vibrations from construction blasting.* The Explosives Engineer. 1960.

Linehan, P. & J.F. Wiss: *Vibration and air blast noise from surface coal mine blasting.* SME-AIME Fall Meeting, 1980.

Lopez Jimeno, C.: *Las Voladuras Submarinas y sus Efectos Ambientales.* I Curso sobre Control de Vibraciones Producidas por Voladuras. Fundación Gómez-Pardo, 1982.

Lopez Jimeno, C.: *Criterios de Prevención de Daños para Vibraciones Generadas por Voladuras y Transmitidas a Travês del Terreno.* E.P.M., 1982.

Lopez Jimeno, C.: *Efectos Desestabilizadores en Taludes Rocosos por Acción de las vibraciones Generadas en Voladuras.* II Curso sobre Control de Vibraciones Producidas por Voladuras. Fundación Gómez-Pardo, 1983.

Lopez Jimeno, C.: *Ajuste Estadistico de Leyes de Propagación de las Vibraciones Terrestres.* I Seminario de Ingenierĭa de Arranque de Rocas con Explosivos en Proyectos Subterráneos. Fundación Gómez-Pardo, 1986.

Lopez Jimeno, C. & M. Abad: *El Seccionado de Cargas: Una Técnica Eficaz para Disminuir el Nivel de Vibraciones.* Canteras y Explotaciones, Noviembre, 1986.

Lopez Jimeno, C. & E.: *Principales Parámetros de Diseño en las Voladuras a Cielo Abierto y su Conexión con los Fenómenos Vibratorios.* Tecniterrae, Septiembre, 1985.

Lopez Jimeno, E.: *La Voladura de Rocas y las Alteraciones Ambientales.* Canteras y Explotaciones, Septiembre, 1985.

Medearis, K.: *Rational damage criteria for low-rise structures subjected to blasting vibrations.* Proceeding Institution of Civil Engineers, Sept. 1982.

Mein-Ban Lo: *Prediction of ground vibration induced by pile driving.* 15th Conference on Structural Engineering. Filipinas, 1981.

Oriard, L.L. et al.: *Short-delay blasting at Anaconda's Berkeley Open Pit Mine, Montana.* AIME, Annual Meeting, Nevada, 1980.

Oriard, L.: *Notes from blast damage orientation.* 1984.

Persson, P.A.: *Underground blasting in a city.* Subsurface Space, 1980.

Phang, M.K. et al.: *Investigations of blast-underground vibrations from surface mining.* The University of Alabama, 1983.

Preston, D.J.: *New methods in production blast monitoring and optimization.* Du Pont Canada, Inc., 1983.

Shoop, S.A. et al.: *Site-specific predictions of ground vibrations induced by blasting.* Annual Meeting of SME/AIME, 1983.

Siskind, D.E. et al.: *Structure response and damage produced by airblast from surface mining.* US Bureau of Mines, RI, 8485, 1980-

Siskind, D.E. et al.: *Structure response and damage produced by ground vibration from surface mine blasting.* US Bureau of Mines, RI 8507, 1980.

Skipp, B.O.: *Blasting vibrations – Ground and structure response.* Dynamic Waves in Civil Engineering, 1970.

Skipp, B.O.: *Ground vibration instrumentation. A general review.* Instrumentation for Ground Vibration and Earthquake, 1978.

Smith, N.S.: *Contribution of sub-grade explosive charge as a source of ground vibrations in bench blasting.* University of Missouri, Rolla, 1979.

Smith, N.S.: *An investigation of the effect of explosive primer location on rock fragmentation and ground vibrations.* University of Missouri, Rolla, 1983.

Stachura, V.J. et al.: *Airblast and ground vibration generation and propagation from contour mine blasting.* US Bureau of Mines, RI, 8892, 1984.

Stagg, M.S. et al.: *Measurement of blast induced ground vibrations and seismograph calibration.* US Bureau of Mines, RI, 8506, 1980.

Stagg, S. et al.: *Effects of repeated blastings on a wood-frame house.* US Bureau of Mines, RI, 8896, 1984.

Walker, S.: *Development of response spectra techniques for prediction of structural damage from open-pit blasting vibrations.* University of Leeds, 1981.

Walter, E.J.: *Decay of seismic pulses near the source.* Earthquake Notes, 1960.

Winzer, S.R.: *Initiator firing times and their relationship to blasting performance.* 20th US Sumposium on Rock Mechanics, 1979.

CHAPTER 34

Flyrocks and their control

34.1 INTRODUCTION

Flyrock, also called rock throw, is the uncontrolled propelling of rock fragments produced in blastings and constitutes one of the main sources of material damage and harm to people.

The conditions which favor flyrock are as follows:

Geology

Intensely fissured and jointed rocks facilitate the appearance of flyrocks more than massive and homogeneous rocks. However, as the latter require large quantities of energy to obtain a good fragmentation, this type of rock usually causes more problems.

Very careful control should be be observed when blasting in karstied ground with a large number of voids and vugs.

Explosives and their distribution

The explosives which have a high Bubble Energy (ANFO, for ex.) produce more rock throw than others which have a more elevated Strain Energy, such as gelatin explosives.

As to distribution, it has to be made certain that the geometric variables of the blast coincide with those of the design, especially in the following cases:

- When the top part of the bench is broken due to excessive subdrilling from the benches above or unsufficient stemming to avoid the risk of crater effect, Fig. 34.21.
- When the face is very irregular, with areas along the length of the explosive column which have very little burden.

The blast design

As indicated in other chapters, flyrock control starts with a correct blast design.

In multiple blastings, apart from inspecting the state of the face of the round and correctly size the stemming, it is fundamental to choose the timing of the stemming between rows, so as not to have too much confinement in the last blastholes which can produce flyrock.

34.2 MODELS TO CALCULATE THE THROW OF FLYROCK

The empirical models proposed by the Swedish Lunsborg and Persson and the American Roth are tools that predict the maximum throw of flyrock.

Below, the most important points of these models are cited.

34.2.1 *Swedish model*

The Swedish Detonic Research Foundation (1975) developed a theoretic model that permits the estimation of the maximum distance reached by a fragment under optimum conditions.

From scaled tests, with high speed photography and theoretical calculations, the following equations are proposed to determine the initial velocity of throw in the blastings where crater effect was produced:

$$v_o = \frac{10\,D \times 2600}{T_b \times \rho_r}$$

where: v_o = Initial velocity (m/s), D = Diameter of the blasthole (Inches), T_b = Size of the rock fragments (m), ρ_r = Rock density (kg/m^3).

By using the standard equations of ballistic trajectory and taking into account that the product $v_o \times T_b \times \rho_r$ depends upon the diameter of the blasthole, the maximum throw length was calculated.

The results obtained are shown in Fig. 34.2, or they can be found analytically from:

$$L_{\max} = 260 \times D^{2/3}$$

$$T_b = 0.1 \times D^{2/3}$$

In practice of bench blastings, it has been proven that the throw lengths are much smaller than when crater effects are produced. Therefore, in well designed blasts, the throw lengths can be calculated from Fig. 34.3. For example, for a specific charge of 0.5 kg/m^3, the maximum throw range would be given by:

$$L_{\max} = 40 \times D$$

and if the blastholes were drilled to 102 mm (4″), it would be:

$$L_{\max} = 160 \text{ m}$$

$$T_b = 0.25 \text{ m}$$

34.2.2 *American model*

This model, owed to Roth (1979), is based upon the

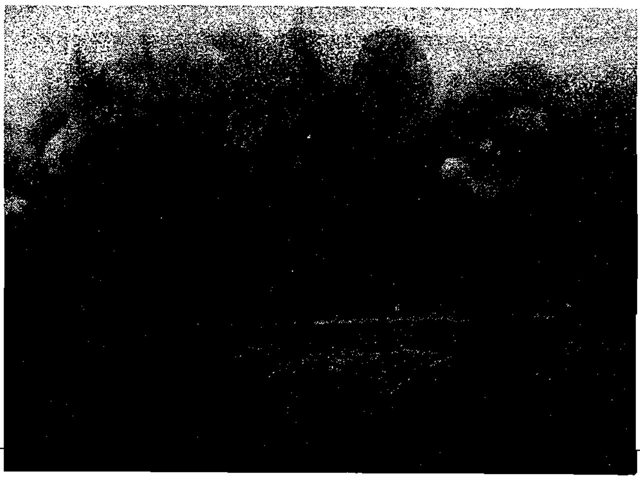


Photo 34.1. Flyrock during the blasting.

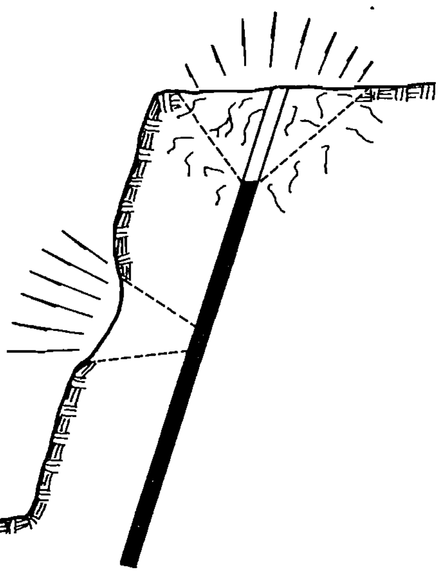


Fig. 34.1. Crater effects that could cause flyrock in bench blasting.

equation proposed by Gurney to calculate the initial velocity of the fragments propelled by an explosive:

$$v_o = \sqrt{2E} \times f(q_l/m_l)$$

where: v_o = Initial velocity, $\sqrt{2E}$ = Gurney's constant, function of the explosive, q_l = Concentration of explosive per unit of length, m_l = Total mass of material per unit of length.

For the flyrock coming from vertical faces the equation has been modified to:

$$v_o = \sqrt{2E'} \times q_l/m_l$$

where $\sqrt{2E'}$ is smaller than $\sqrt{2E}$ as the direction of detonation is tangent to the rock. The author suggests taking $\sqrt{2E'}$ = *VD/3 for many explosives, where VD* is the detonation velocity. For ANFO, the value of the radical is 0.44 *D*.

If the energy losses are taken into account, the previous equation is transformed into:

$$v_o^2 = 2 \times E' \times \left(\frac{q_l}{m_l}\right) \times$$

$$\left[1 - \frac{k_1 \times E_s + K_2 \times E_j}{E'}\right] - 2 K_3 \times E_r$$

where: E_s = Seismic energy generated per unit weight of explosive, E_j = Energy to crush a unit weight of rock, E_r = Energy absorbed to fragment a unit weight of rock, K_1, K_2, K_3 = Proportionality constants.

The equations of v_o^2 expressed in (m/s) for different types of rocks are transformed into:

Granite

$$v_o^2 = 3{,}487 \times 10^6 (q_l/m_l) - 584$$

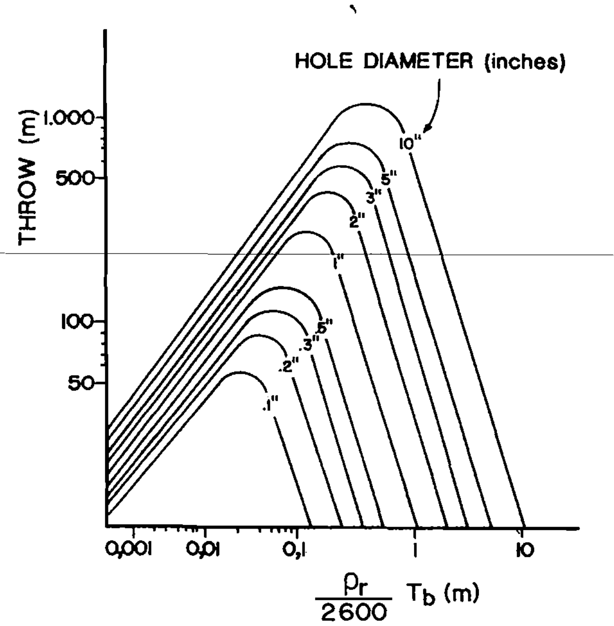


Fig. 34.2. Calculated maximum throw versus boulder size with blasthole diameter as a parameter (Lundborg et al.).

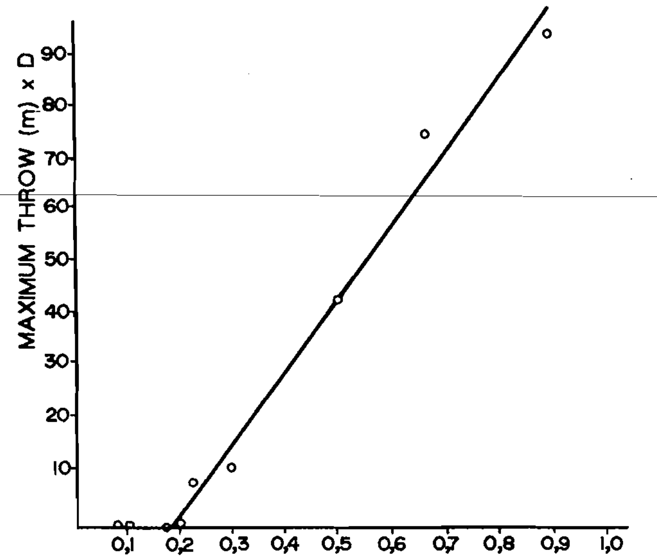


Fig. 34.3. Maximum throw length as a function of specific charge.

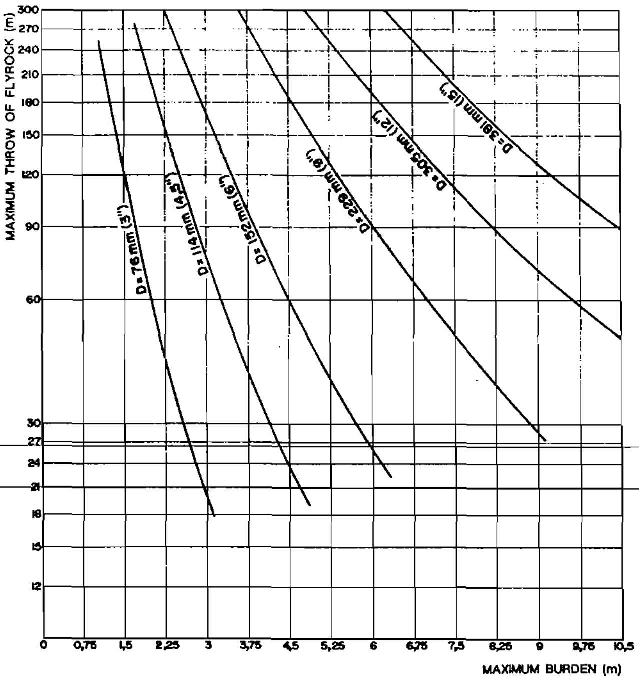


Fig. 34.4. Maximum range of vertical face flyrock from ANFO loaded shots in limestone.

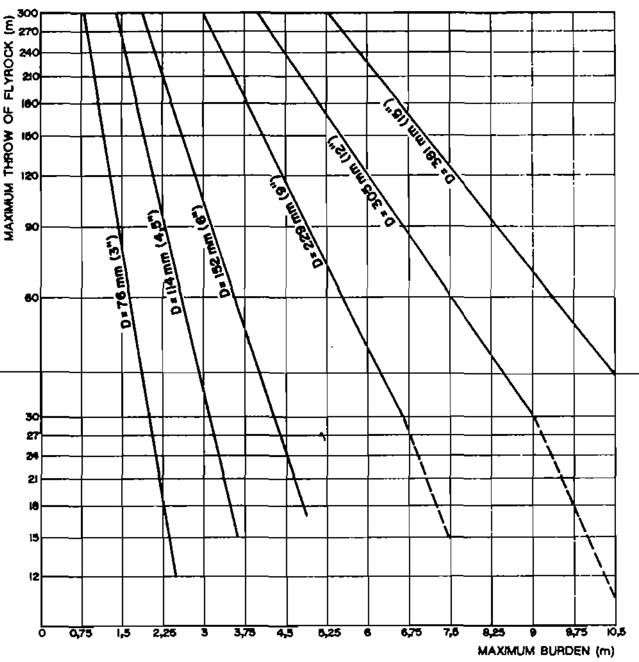


Fig. 34.5. Maximum range of vertical face flyrock from ANFO loaded shots in granite.

Limestones and dolomites

$$v_o = 3 \times 10^6 (q_l/m_l) - 200$$

Going back to the ballistic trajectory formulas, the theoretical maximum ranges for a single blasthole can be estimated.

For flyrock coming from a free face, the estimations can be based upon the nomographs of the Figs 34.4 and 34.5. Knowledge of the type of rock, the diameter of the blasthole and the type of explosive are a requirement. As these nomographs were determined for ANFO, if water-gels are used the distances should be increased by 50 %. The burden value should also be corrected if cavities or rock loss exist on the free face from previous blastings.

For flyrock from the bench tops, an empirical approximation is proposed, based upon the reduced depth $h/Q^{1/3}$, where h is the depth of the end of the charge and Q is the total quantity of explosive, Fig. 34.6.

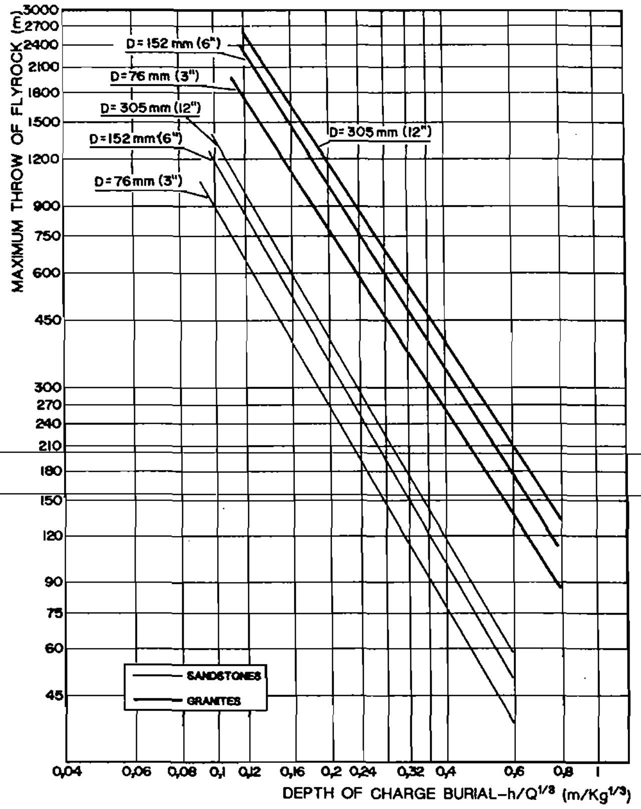


Fig. 34.6. Maximum range for bench top flyrock for ANFO loaded shots in granite and sandstone.

34.3 COVERINGS

Coverings are all the elements used to cover the blastings in order to avoid rock throw or any other material that could harm people, buildings etc.

Generally speaking, any protection system should comply to the following characteristics:

- Reduced weight and high resistance.
- Ease of union or overlaping of the elements
- Permeability to gases.
- Ease in placing and removing.
- Economical and reusable.
- Good size to cover large areas, etc.

According to the type of blast, different coverings will be used.

34.3.1 *Ditch blasting and excavation of lots*

When blastings are carried out in small ditches and inhabited areas are nearby, a covering of loose sand can be used with thicknesses equal to the stemming height, maintaining a minimum of 0.8 to 1 m, Fig. 34.7.

Owing to the weight of the sand, the explosive charges should be slightly larger than in unprotected blastings.

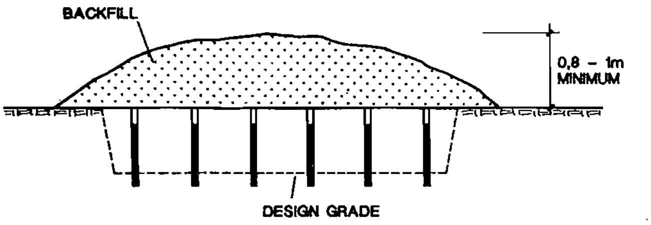


Fig. 34.7. Protection of a ditch blast by means of a sand covering.

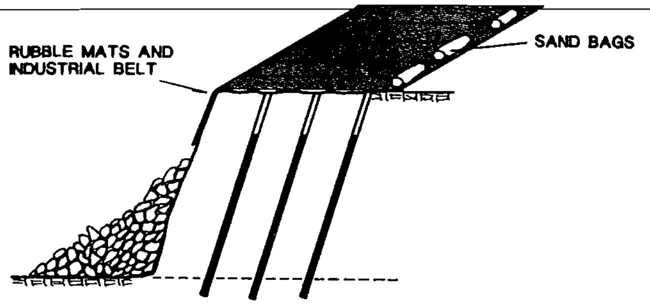


Fig. 34.8. Protection of a bench blast.

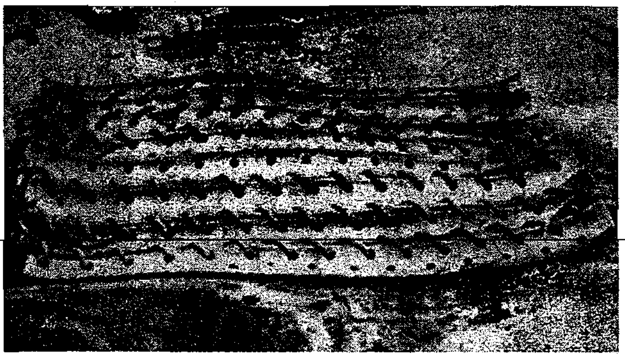


Photo 34.2. Blasting mat.

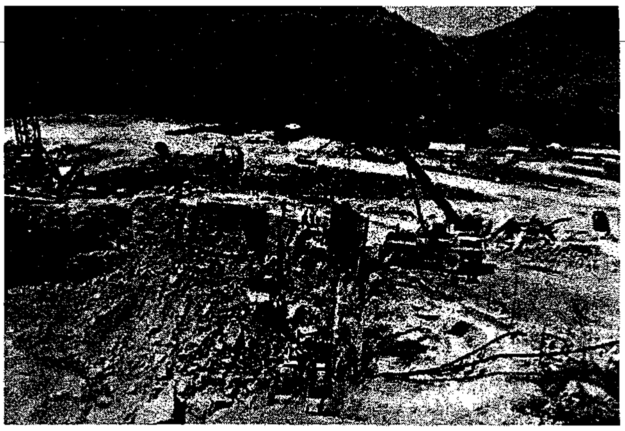


Photo 34.3. Placing of a heavy covering upon a small blast.

Another system consists in overlaping conveyer belts and pinning them down to the ground with sandbags, for example. At the same time metal screening or mesh, nylon nets, or rubber tires that overlap, etc., can be used.

In lot excavations with explosives, the most common system is that of the conveyor belts. These should cover the horizontal surface of the round as well as the free bench face, Fig. 34.8.

In all instances it is necessary to make certain that the connection circuits are all right before and after the coverings have been placed.

34.3.2 *Secondary blastings*

Secondary blastings are a common source of flyrock. In order to control these, besides using the protection systems mentioned, it is recommended that the boulders be removed to areas where they do not disturb the operation, and that the blasts be sufficiently closed in by the slopes of the exploitation to eliminate part of the noise produced by the secondary blast and, at the same time, take advantage of the shielding effect of the faces with respect to the fragments of flying rock.

34.3.3 *Demolitions*

In demolition work, the blastholes drilled in the exterior structural elements should be protected by heavy screens made up of hanging conveyor belts. Special pistols are used to nail them in place, and underneath the holding points there should be sufficient space to allow the gases to escape because, if this is not done, the protections would be torn down in the first blasting.

Other types of complementary protections are metallic screens and bales of straw.

On the other hand, as the lower parts of the structures are usually not protected, it is necessary to close all the door and window openings to avoid rock throw from the interior. In these operations, heavier materials are used such as wooden boards, metal plates, sandbags, etc., which should be installed before charging the blastholes to eliminate possible damage to the circuit lines of the blast. Occasionally, the whole perimeter of the structure to be demolished is covered with geotextile sheets which act as complementary protection.

34.3.4 *Safety area from which the round is fired*

In any surface operation, during the blasts there is always a desired displacement of the muckpile, a normal rock throw distance, and a safety area around the blast. The size of these zones depends upon the characteristics of the blasting, making them vary from blast to blast. However, the prediction models can serve as a tool to define these three areas, Fig. 34.9.

The rock throw that falls in the safety area, farther away than normal, as well as that that surpasses it should be studied in order to establish their origin and the corrective measures to be taken.

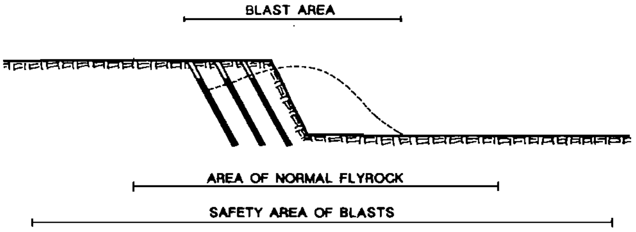


Fig. 34.9. Areas around the blasts in function with rock throw.

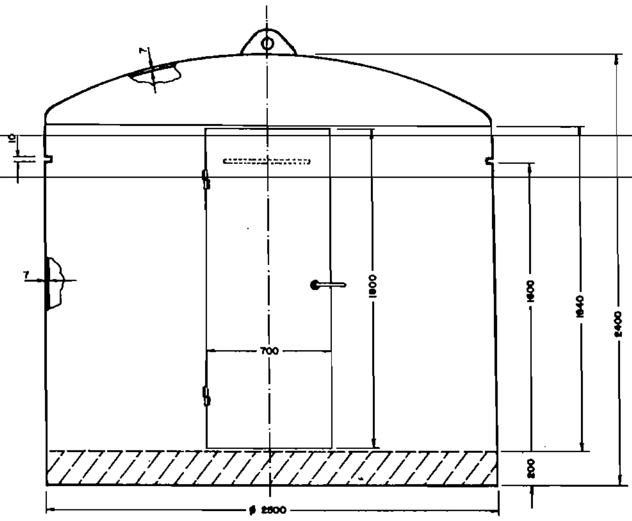


Fig. 34.10. Blasting shelter to protect the shot firer.

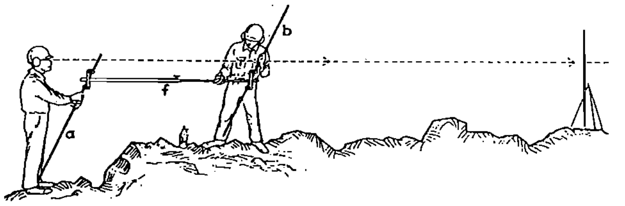


Fig. 34.11. Marking out of a blast in irregular ground.

As to where the shot firer should be placed to fire the rounds, he should be outside the safety areas and use some system of protection such as a metallic blasting shelter, Fig. 34.10, nearby underground operations, front end loader shovels, etc.

34.4 RECOMMENDATIONS FOR CARRYING OUT BENCH BLASTINGS

In order to control rock throw in bench blasts, apart from the adequate protection measures, the following recommendations should be followed:

Photo 34.4. Marking out the collaring points in a large diameter blast.

- Perfect marking out of the drilling patterns, especially in ground with an irregular profile, Fig. 34.11.
- Control of the deviations and depths of the blastholes.
- Burden size for the blastholes of the first rows.
- Check for vugs in the rock mass.
- Control of the charging of the explosive and its distribution along the length of the blasthole.
- Careful stemming, measuring its height and using the proper material.
- Selection of an initiation sequence that gives good break direction to the blast.
- Initiation in the bottoms of the holes.

REFERENCES

Bergeron, M.: *Controle des Roches Volantes a la Mine Jeffrey.* SEEQ, 1986.

Lopez Jimeno, C.: *Estudio de la Excavacion de una Central Térmica.* Proyecto Fin de Carrera, ETS Ing. de Minas, Madrid, 1978.

Lundborg, N. et al.: *Keeping the lid on flyrock in open-pit blasting.* E/MJ, May, 1975.

Melnikov, N. & M. Chesnokov: *Safety in open-cast mining.* MIR Publishers, Moscow, 1969.

Roth, J.: *A model for the determination of flyrock range as a function of shot conditions.* Bureau of Mines, Open File Report, 77-81.

Tamrock: *Handbook on Surface Drilling and Blasting,* 1984.

CHAPTER 35

Safety measures for drilling and blasting operations

35.1 INTRODUCTION

In order to carry out drilling and blasting under safe conditions, the following aspects must be observed:

1. Comply with the Rules and Regulations that are in effect.
2. Proper technical instruction for the operators, blasters and personnel who handle explosives.
3. Machinery, explosives, accessories and initiation systems must be used under safety conditions.

The drilling superintendent should supervise these three conditions as, if not, the risk of accident will increase owing to over confidence, distractions, lack of knowledge and non-compliance with the safety rules, etc.

In this chapter, a general guide of basic recommendations is given which obviously should be complemented with the existing legislation.

35.2 BLASTHOLE DRILLING

35.2.1 *General safety measures for blasthole drilling*

The operation of drilling implies following a series of safety measures in order to minimize potential hazards to people as well as to material objects.

Drilling will be carried out according to the existing rules or policies, either official or those set by the company.

The operators should have received proper training and have studied the instruction book for the machine or machines which they are to handle, Fig. 35.1.

The members of the drill crew should be given garments which provide adequate protection (helmets, boots, gloves, glasses, masks, etc.), and use clothing and accessories that are not loose so as to avoid their catching on the moving parts of the machine.

The personal protection objects and those for the machine should be in good condition; if not, do not commence drilling.

The protection systems for the machine should not be disconnected, in order to avoid damage to itself or to people.

The starting and manuevering controls should be protected so as to avoid manipulation by other people, which could constitute a risk.

The compressor on the rig should be equipped with a fire extinguisher and a first aid kit, which the operators must know how to use.

If the work conditions are poor or dangerous, the equipment should not be used.

Place warnings on the control pannel to advise of these conditions.

There should be signs that are well visible advocating the necessity of personal protection, Fig. 35.2.

35.2.2 *Safety precautions before starting equipment*

The crew members should be prepared to assume possible risks and have the means to confront them, as well as knowing where to look for help.

The driller should check the whole rig, even if everything was working correctly in the previous shift.

The drill crew should inspect the premises where they are going to work, its potential limitations, as well as the accesses to the area.

The pressurized hoses will be securely anchored, especially the main hose, which should have an additional safety cable at the connection point.

The threads and connection elements must be correctly tightened.

Check all fluid levels, oiling points and cleanliness of the machine according to the manufacturer's instructions, and make certain that all tools and equipment are in proper places and in good condition.

Possible fuel and other fluid losses must be watched, and the deposits will be purged according to the service instructions.

35.2.3 *Safety measures during starting*

When starting the machine, the following precautions should be observed:

– Make certain that unnecessary personnel are not on the rig or in the surroundings.

– Check to see that all controls are in the correct position.

– Inspect any possible warning signs or instructions on the rig.

– Start the drill by authorized operator, from the proper position and in the open air or with good ventilation.

– Never leave the rig when it is running.

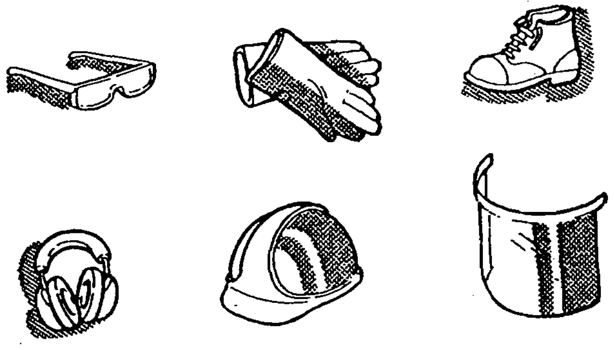


Fig. 35.1. Safety garments for personal safety.

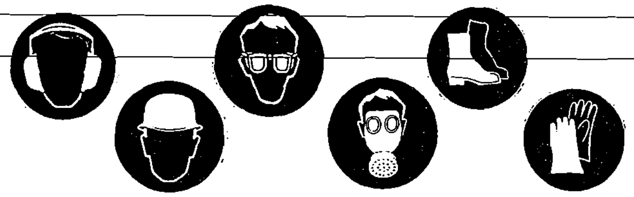


Fig. 35.2. Signs that warn of the obligation to use safety material.

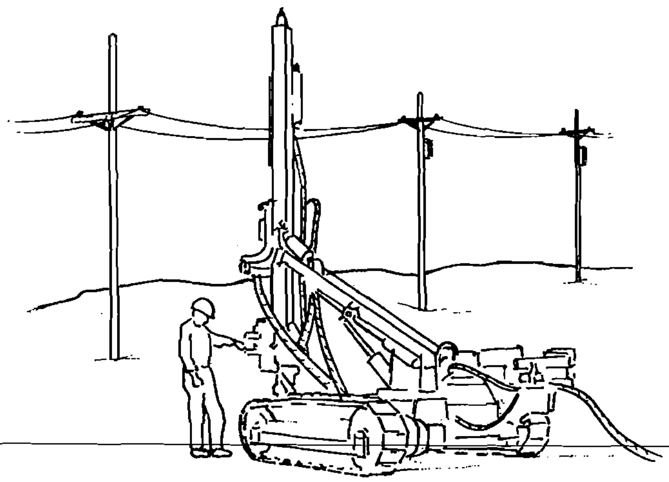


Fig. 35.3. Danger of contact with power lines.

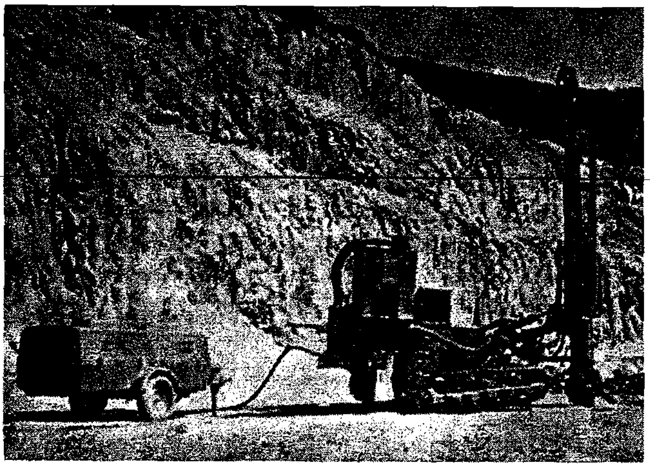


Photo 35.1. Drill equipment with compressor.

35.2.4 *Safety measures after starting*

Check proper functioning of all controls. Pay attention to any unusual noises.

35.2.5 *Safety measures for surface movements of the rig*

Before moving the equipment, the following should be checked out:

Make certain that the ground is in good condition for safe moving. If not, it must be prepared with the available auxiliary equipment such as tractors, shovels, etc.

The existance of elevated, surface or underground power lines. The rig must be kept at a minimum safety distance of 10 m from any electrical line, Fig. 35.3.

See if there are any underground pipelines, for gas or water, or telephone lines, in the way.

Stability of the slopes near the work area.

Slopes of the cuts where the rig is working. If necessary, the machines can be tied with cables and wenches.

The drilling accessories, especially drill pipes, must be immobilized.

The feed or drill mast must be put down when moving, 35.4.

When moving, the operator occupies the driver's seat designated by the manufacturer. No other person will be authorized on the rig or between the compressor and rig when it is being towed, 35.5.

The rig will not move over previously drilled areas.

The slopes encountered during movement must be within the limitations set by the manufacturer.

When the translation chains, hammer advance chain and drill pipe chain are in movement, the operators should remain at a safe distance.

When working with auxiliary electric equipment, a helper is needed for maneuvering between blastholes. He must control the cables, avoid passing over drilled holes, help direct the machine so that it won't go near the edges of the slopes and make certain that the hydraulic jacks are out of the way.

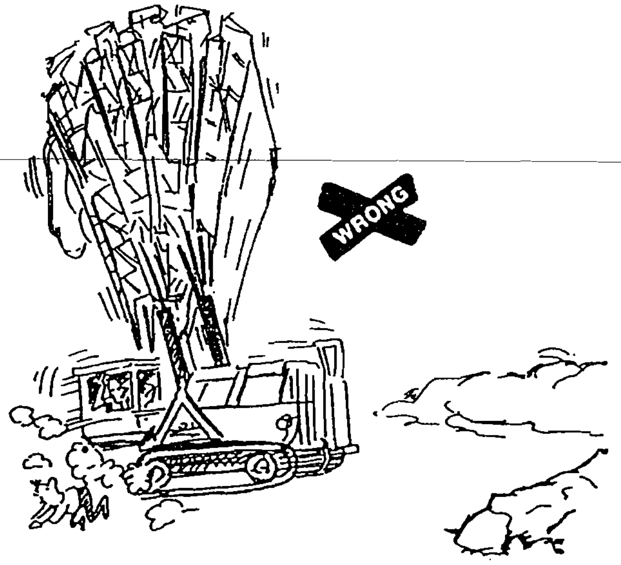


Fig. 35.4. Incorrect position of the mast when in movement.

35.2.6 *Safety measures when working underground*

Be sure that the roof of the drift is in good condition and, if not, take steps to make it safe, apart from using proctective cabins on the rig, Fig. 35.6.

Make certain that there is sufficient ventilation for work at the far end of the drift, considering that each miner requires 40 liters per second and 180 liters per horsepower and second when working with diesel equipment.

The floor over which the traffic will flow must be cleaned of debris and other materials.

Check the electric lines, compressed air and water pipes, and make certain that they a properly anchored to the walls in access ways.

The work area must be properly illuminated.

Grounding must be available, conbined with cut-off devices and alarms in the electric machinery.

The waste water drains must be kept clean and all water removed from the work zone.

All necessary work utensils must be kept in order and in their proper place.

The bottom of the blastholes that are unservable must be visibly indicated and no other hole drilled within a distance of 20 cm. Never use blasthole bottoms as collaring points.

35.2.7 *Safety precautions during drilling*

When the rig is being placed, the ground irregularities or unstableness must be taken into account, or the presence of underground work or pipelines. Make certain that the necessary rock mass for protection exists in accordance with the static and dynamic characteristics of the machine, Fig. 35.7.

In steep ground, when working with portable compressors they must be kept in a safe place.

Once the rig has been leveled and immobilized, the mast or drill tower can be positioned, slowly and paying attention to any obstruction that could exist.

Any maneuver that is potentially unsafe will require the aid of a helper who should be in visual contact with the operator.

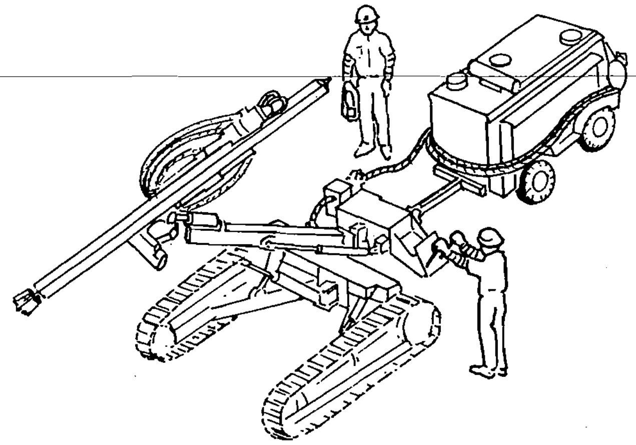


Fig. 35.5. Proper position of the operators during transport of the drill rigs.

Fig. 35.6. Checking wall and roof conditions.

The drilling site should have appropriate conditions of visibility, as much for the operators as for other crew members.

Never collar upon bottoms of old blastholes. Don't use the lifting and lowering mechanisms of the rock drill for any other purpose than intended.

In the machines that have automatic rod or drill pipe changers, the operator should frequently check the mechanisms that make the drilling accessories run and stop.

When changing the drill pipes, attention should be paid to the drilling accessories (shanks, coupling sleeves, etc.) which could be poorly connected.

The drilling accessories must always be in good condition. The pieces which appear worn could affect the safety of the operation and should be thrown away.

The drilling accessories should be stored in adequate places, protected from dust and blows. Never hit metal against metal without eye protection, Fig. 35.8.

During drilling, the machine must have all its control levers, protections and guard plates in perfect condition.

When drilling, write down the numbers indicated on the controls and also note in the report any incidents which might arise.

The crew members should never go under the rotary drills when the jacks are lifted, unless fixed stops have been placed.

When a drill pipe is taken out of the carousel, make certain that it is correctly directed.

When drilling blastholes, check the descent of the rotary head or percussion hammer.

Operators should always remain away from moving parts of the drill, such as hammer chainfeeds, cables, belts, compressor, etc., and the levers must be operated from correct positions.

The drill steel, couplings, bits, etc., that have just been used should not be touched with bare hands as there is risk of burns.

When heavy accessories are lifted or handled, the following precautions should be taken:

Keep legs open, placing a foot on each side of the object.
Bend the knees and squat down, keeping the head erect.
Grab the object with the whole hand, embracing it with the arms. Keep weight on both feet, lifting the object with the back muscles of the legs, Fig. 35.9.

When placing the object, don't turn the body and keep it close to where the object is to be deposited.

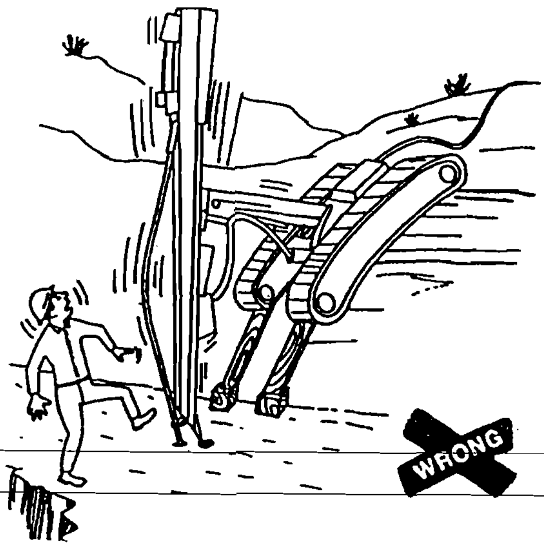


Fig. 35.7. Never work with the equipment when it is unstable or in a precarious position.

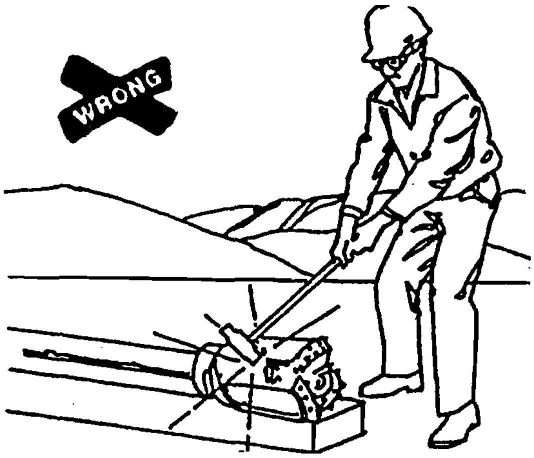


Fig. 35.8. Never strike metal objects against each other without eye protection.

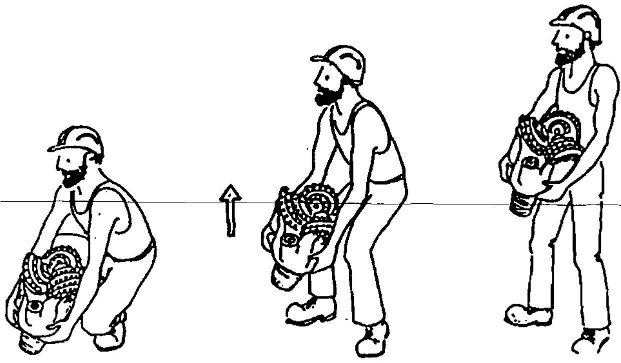


Fig. 35.9. Every precaution should be taken when lifting heavy objects.

35.2.8 *Safety measures after drilling*

Never leave the machine with the motor still running.

The procedure to stop the machine should be carried out according to the manufacturer's instructions.

Never park the machine in areas that could become flooded, or where it could be reached by flyrock, Fig. 35.10.

If there is any circumstance that could affect the use of the rig, a warning note should be left on the ignition controls before leaving the machine.

Try to avoid parking on a slope. If necessary, use the prescribed braking apparatus and place proper wedging.

Before leaving the rig, release pressure from all circuits, leave controls on off and park positions, using the existing brakes and taking away the keys, if they exist.

35.2.9 *Safety precautions in maintenance and service*

The personnel which is to handle the maintenance, repair and service operations will be designated by the Company.

The service instructions specified by the manufacturer in its Service and Maintenance Manual will be strictly followed at all times.

During maintenance and repair, the rock drill must be perfectly braked and immovilized so that it can not move unexpectedly.

The starting controls should be blocked and marked so that only authorized persons can use them.

Any operations that imply movement of the drill feed or mast must be carried out with the driller in the control cabin, and any other person nearby must place himself behind the machine.

Servicing of the motor compressor batteries implies potential risks of burning by the sulphuric acid, and fires and/or explosions, as they produce hydrogen and oxygen.

The personnel attending to the batteries must always wear safety glasses, gloves and clothing that is acid resistant, Fig. 35.11.

When changing a battery, the ground terminal will be the first to be disconnected and the last to be connected.

The electrolyte level should be checked frequently and when necessary add distilled water, always before starting the motor, never after stopping. A correct level means lower volume of gases inside the battery.

Never use a flame to check the level of a battery, always use a flashlight or portable lamp.

When charging the batteries, leave the caps off the deposits. The terminals from the charging aparatus should be applied and removed with the switch turned off.

Never smoke near batteries or when working with them.

No crew member is allowed to climb up the drill tower unless it is a case of extreme necessity. If this case arrises, a safety belt that is attached to the tower must be worn.

Hands, arms and the rest of the body, as well as work clothes, must be kept away from any moving part of the rock drill or compressor (chains, pulleys, drill, etc.).

Do not open any deposit, water or oil hose when the rig is working, or if they are pressurized.

Check the state of the safety valves at least once a week. They should be in perfect working condition.

Never repair safety valves of the pressurized circuits. Always replace with new ones.

Never go above the pressure recommended by the manufacturer for any pressurized equipment.

Do not use air hoses with pressures over 0.2 MPa for cleaning filters, work clothes, dust, etc. When necessary, use protective glasses, Fig. 35.12.

Refuelling must be carried out with the motors off and in perfectly ventilated areas.

Avoid spilling fuel on surfaces that are at higher than environmental temperatures. The refuelling hoses should have appropriate nozzels.

When fuel is spilled, before starting the engine clean all affected surfaces.

Smoking is forbidden in a radius of at least 10 m from the refuelling point, as well as any flames, incandescent materials or anything that can produce sparks, Fig. 35.13.

Have fire extinguishers available at the refuelling point for type B fires (grease, gasoline, dissolvents, paint, etc.).

Avoid completely filling the fuel tanks as their volume can vary with the temperature.

When checking the level of the cooling liquid in the motor compressor, the engine must be off and the radiator at environmental temperature.

35.3 BLASTINGS

Before indicating the recommendations for the different stages of blasting, it must be stated that, except in underwater blasts, *it is expressly forbidden to charge blastholes during drilling*. There could be accidental detonation through contact with the drilling tools and equipment.

35.3.1 *Measures to be taken when storing explosives*

Always store explosives in powder magazines that fulfill the acting rules and regulations.

Photo 35.2. Surface powder magazine with natural protection.

Make certain that the magazine is clean, dry, well ventilated, reasonably cool, of solid construction, fire resistant, and securely locked.

Always use or give out the oldest products first or, which is the same, in the order that they entered the magazine.

Store the products of the same type and class in such a manner that their identification is simple. This entails their revision and age control.

Take special precautions with defective boxes or broken wrappings. Place them in a separate area of the magazine.

Place the magazines in isolated and strategic locations, in accordance with the distances set by the regulations in force.

Consult with the manufacturer as to cleaning procedures when any liquid substance of deteriorated explosives has spilled on the magazine floor.

If artificial illumination is necessary, use safety lamps.

Immediately repair any leaks that might appear from walls or roof.

Never open or rewrap the boxes of explosives inside the magazine.

Do not leave loose explosives or open boxes inside the magazine.

Never store blasting caps or other initiation accessories in the same magazine.

Do not store detonating cord in the same place as the electric blasting caps.

Do not store metal objects that could produce sparks in the magazine.

Never store oils, gasolines or dissolvents in the same area.

Matches or lighters are strictly forbidden inside or near the magazine.

Place clearly visible warning signs around the installations and when transporting explosives, Fig. 35.14.

35.3.2 *Precautions when transporting explosives inside the working area*

Strictly obey the conditions established by the Regulations in force.

Make certain that any vehicle assigned to the transport of explosives complies with the stipulated regulations.

Check the running condition of the vehicle to see if everything is in good order. Do not allow the load to stick out and have canvas on hand to cover the explosives if it rains.

Carry fire extinguishers on the vehicles, properly placed and in easy access. The knowledge of their use is obligatory for drivers and helpers.

The engine of the vehicle must always be turned off when loading and unloading explosives.

Check the floor of the vehicle to be certain that there are no cracks or holes.

Always load and unload the vehicles during daylight hours and never when there are electric, sand or snow storms.

When loading and unloading explosives, only autho-

Photo 35.3. Flexible measuring tape to control charge height and blasthole depth.

rized personnel are allowed to remain in the area, prohibiting any other activity within a radius of 50 m.

Never transport metal, fuel or corrosive materials with explosives.

Smoking is strictly forbidden for anyone in or near the vehicle.

Do not allow the boxes of explosives to be opened when still in the vehicle or in the unloading area until this operation is finished.

Never transport blasting accessories with explosives. Detonating cord is considered an industrial explosive.

Always transport the explosives in their original wrappings and boxes, or in specially prepared cartons.

Name a person to be responsable for the movement and dispatch of explosives and blasting accessories.

When unloading, explosives, blasting caps, detonating fuses, etc. should be handled with extreme care and never be dropped or receive blows, Fig. 35.15.

Distribute the explosives to be used in the blast in order to avoid leaving large amounts piled up.

Always use routes with the least traffic of personnel, as well as of other machinery.

Place the blasting caps far away from the charging zone.

Supervise unloading of explosives until they are placed in the blastholes and the round connected.

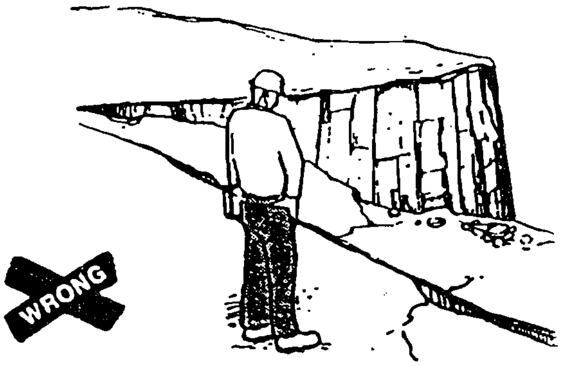


Fig. 35.10. Avoid parking rigs near the walls.

Fig. 35.11. When handling batteries, all pertinent precautions must be taken.

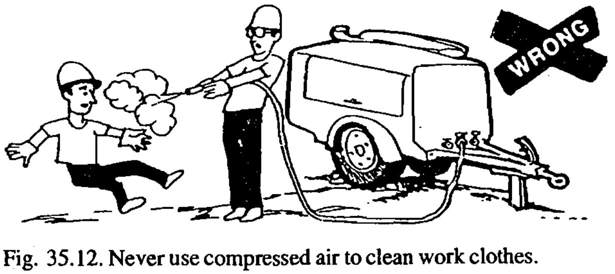


Fig. 35.12. Never use compressed air to clean work clothes.

35.3.3 *Precautions in the blasting area*

Clean the blast area, removing loose rocks, bushes, metals and other materials.

Mark the blasting area limits with sticks or brightly colored flags and prevent the passing of any machinery over it, Fig. 35.16.

When changing shifts, inform the operation personnel of the blasts to be carried out that day.

Prevent access of anyone not directly involved with the handling of explosives to the marked zone and its proximities.

Reduce the number of charging crew members to a minimum and name a person to be in charge and a supervisor.

Correctly mark the location of all blastholes.

35.3.4 *Measures to be taken when preparing the primer cartridge*

Prepare the primers according to the methods recommended by the manufacturers of explosives and check that the initiator is well placed inside the cartridge.

Be certain that during charging there is no tension in the detonator cables or in the detonating cord and in the points of union.

Insert the detonators inside a hole made in the primer cartridges with a special tool called a pricker, which can be of wood, copper, bronze or any metal alloy that does not produce sparks.

The primers should not be prepared much in advance nor in greater quantity than what is to be used immediately. This operation should not be carried out in the magazine nor near explosives.

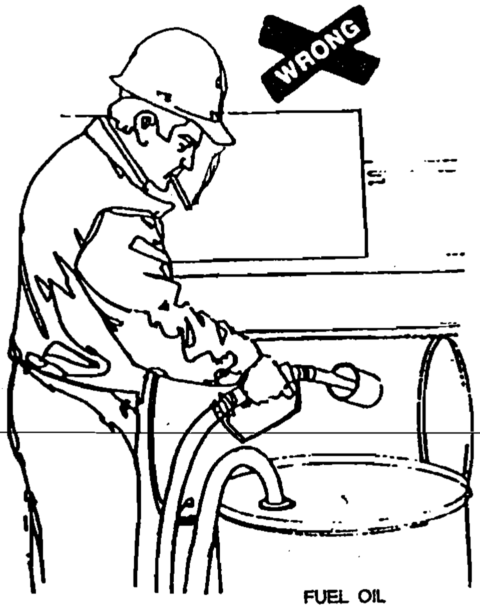


Fig. 35.13. Smoking is forbidden during refuelling.

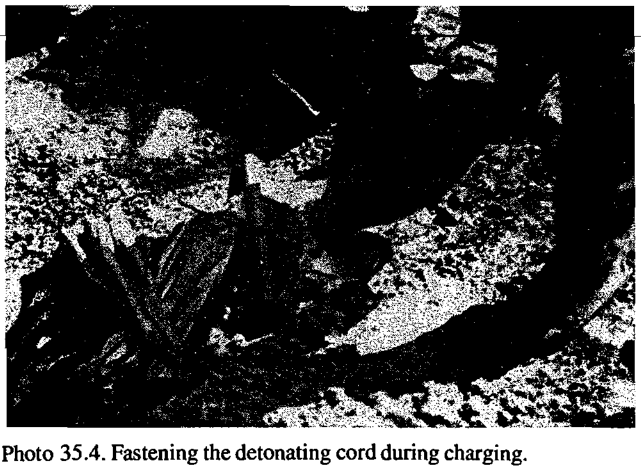


Photo 35.4. Fastening the detonating cord during charging.

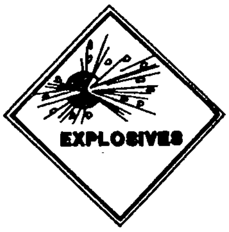


Fig. 35.14. Warning sign for explosives.

35.3.5 *Precautions when charging blastholes*

Examine each blasthole carefully before charging to know its length and condition, using a wooden rod, a scraper or even a measuring tape, Photo 35.3.

Always forsee the possibility of danger from static electricity when using an pneumatic charger and take all necessary precautions such as connecting a ground line. Remember that relatively low humidity in the atmosphere increases the risk of static electricity.

Cut the necessary detonating cord from the reel once it has been inserted in the blasthole and before inserting the rest of the explosive charge, Fig. 35.17.

Prevent the personnel dedicated to charging from exposing part of their body over the blasthole that is being charged or from placing themselves in its same direction.

Wrap the end of the detonating cord around a wooden stick or rock to keep it from falling inside the blastholes, Fig. 35.18 and Photo 35.4.

Waterproof the ends of the cord in blastholes with water.

Check the rising of the bulk explosives and take the necessary precautions in case of holes or cavities in the blastholes which could have been detected during drilling or even when charging, Fig. 35.19.

Never leave surplus explosive in the work area during and after charging the holes.

Do not charge the holes right after drilling without first checking to see if they are clean and free of metal pieces or leftover hot accessories.

Do not deform, mistreat or let the primer drop into the blastholes. Never let any heave weight fall on them, Fig. 35.20.

Never recharge holes that have already been charged and fired before.

35.3.6 *Precautions when tamping*

Confine the explosives in the holes with sand, earth, or any other appropriate incombustible material.

Do not use any type of metal tamping tools. Always use wooden tools or others of adequate material, without metal parts except for the special connectors in articulated tamping rods that are made of a metal that does not give off sparks, Photo 35.5.

Stem with care so as not to harm the initiating accessories, detonating cord, etc., and make certain that they have no knots or kinks.

Never directly tamp the primer cartridges.

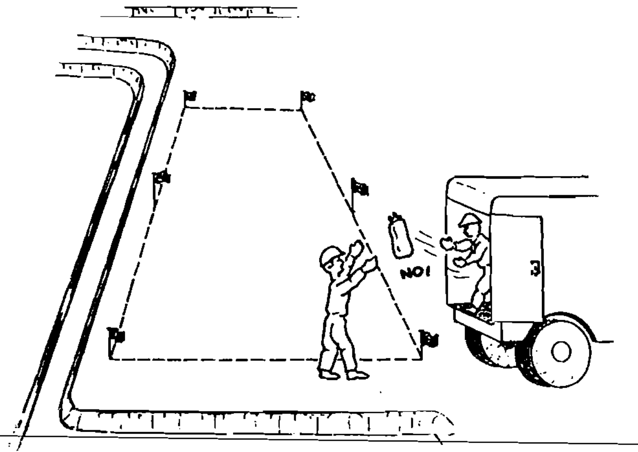


Fig. 35.15. Unloading explosives and blasting accessories should be carried out with great caution to avoid blows.

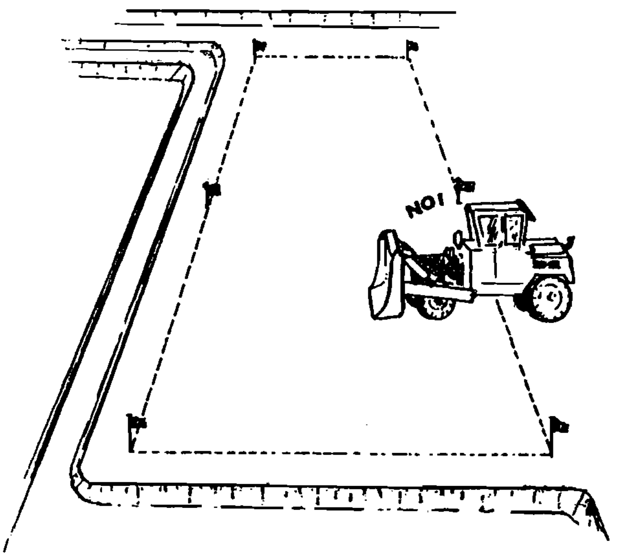


Fig. 35.16. Prevent the passing of machinery over the area of the blast from the moment the preparations are started.

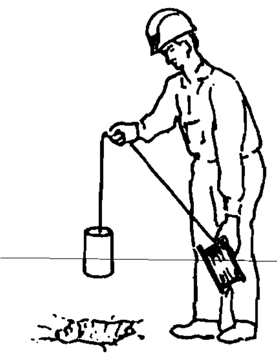


Fig. 35.17. Introduction of the primer with detonating cord in the blastholes.

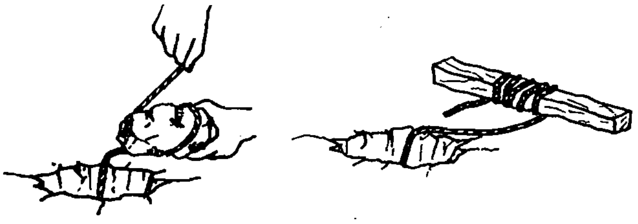


Fig. 35.18. Wrapping of the end of the detonating cord to keep it from falling inside the hole.

Make certain that the stemming material contains no rocks or other objects.

In blastholes with presence of water make certain that the stemmings have not descended before connecting the circuits. If so, correct them.

35.3.7 *Measures to be taken when blasting with electric blasting caps*

Keep the electric blasting cap wires or conductors short circuited and never connect one cap to another until ready to fire.

Check all the blasting caps, one by one or connected in series with the proper blasting ohmmeter designed for this purpose. If checked individually, place yourself in a safe area and introduce the cap in a protective tube, Fig. 35.22.

Never throw the wires in the air to straighten them nor tense them.

Never unroll the leg wires or use the electric caps during a storm or near any source of static electricity or unusual currents.

Never use the caps nor unroll their leg wires near radio transmitters or television stations, electric lines, etc., unless at a safe distance and complying with the regulations in force.

Do not place cables or electric lines near the blasting caps, nor any other explosive until the moment of firing the shot and only for this purpose.

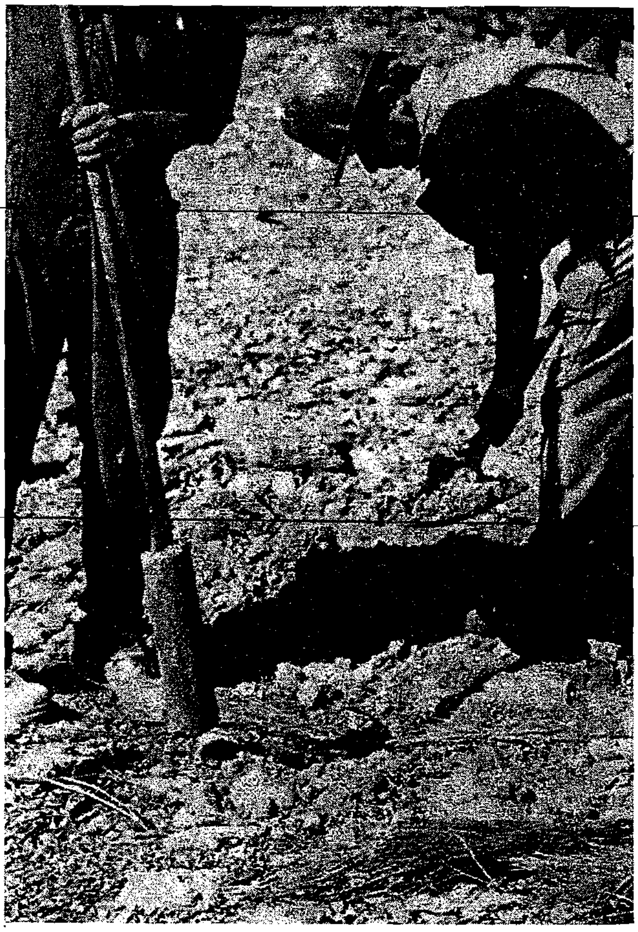


Photo 35.5. Tamping blastholes.

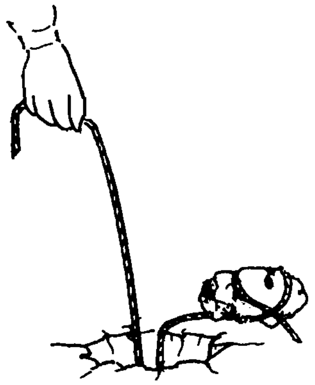


Fig. 35.19. Check the rising level of the explosive when charging.

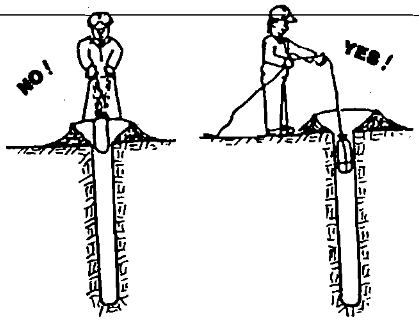


Fig. 35.20. Never let the primer cartridge drop into the blastholes.

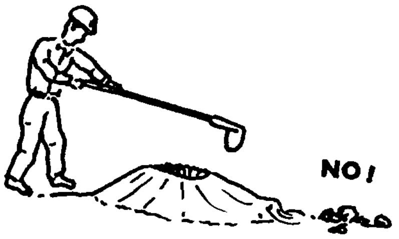


Fig. 35.21. Avoid rocks in the stemming material.

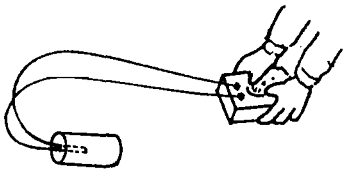


Fig. 35.22. Checking electric blasting caps.

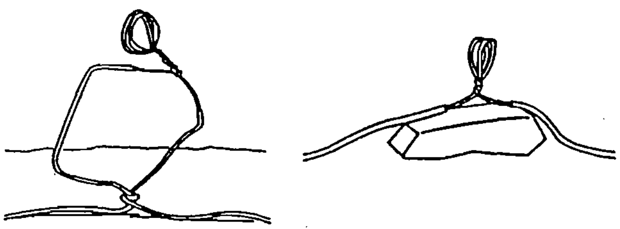


Fig. 35.23. Isolate the connections from the ground.

Take away all surplus explosive from the shot area before the placing or connecting of the caps by the shotfirers.

Push a copper rod into the ground close to the shot before any connecting is done. This is for the discharge of static electricity that could be on the personnel who handle the caps.

Prevent the passing of machinery and the laying out of cables that provide power for them anywhere near the shot area.

Check to make certain that the ends of the leg wires are absolutely clean before making any electric connection.

Carry out the connections as quickly as possible and in one operation, preparing in advance all necessary items.

Once the connection has been done, protect the ends of the wires by completely isolating them so as to avoid any contact with the ground, Fig. 35.23.

Never use blasting caps of different sensitivities on the same circuit.

Avoid proximity of the firing line to other power lines, as well as contact with metal elements, Fig. 35.24.

Before connecting, check the isolation of the firing line and the uniting points of the blasting caps. Use quick connectors for the latter when necessary.

Do not connect the firing line to the blasting machine until ready to fire the blast and keep it short circuited.

Never try to fire a blast with a higher number of caps than recommended by the manufacturer of the blasting machine.

Check and revise the blasting machine periodically as well as the checking equipment.

In the case of cap misfire, never try to open or dismount it.

35.3.8 *Precautions to take when firing with fuse detonators and safety fuse initiation*

Blast as little as possible with fuses and reduce the number of blasthole in each round.

Handle the fuse with care to avoid damaging the covering.

Light the fuse with a fuse lighter designed for this purpose.

Always work with fuses of more than a meter and a half in length. The burning time must be known and make certain that there is time to reach a safe place after lighting. A test fuse can be used for this information.

Cut the fuse when ready to insert it into the fuse detonator. Cut an inch or two off to insure a dry end.

Cut the fuse squarely across with a fuse cutter designed for this purpose and insert it gently against the powder charge of the detonator. Once placed, avoid twisting it.

Crimp the end of the fuse detonator with a cap crimper where the fuse enters. Make certain that the detonator is securely crimped to the fuse to avoid its breaking loose or getting wet.

Never light the safety fuse without covering the explosive sufficiently to prevent sparks from coming into contact with it.

Never have explosives in your hand when lighting the safety fuse.

Before returning to the cut, count the number of holes that have exploded and wait a half hour if there has been a misfire.

35.3.9 *Safety measures before and after the blast*

Make certain that all surplus explosives are in a safe place and that all persons and vehicles are at a safe distance and well protected.

Prevent access to the blast area with adequate personnel and means, Fig. 35.25.

Never fire the shot without a signal of authorization from the person in charge and without having given adequate warning.

Shoot from safe places such as behind protective blasting shelters, loader shovels, etc.

Never go back to the blast area until all smoke and gases have dissipated.

Do not investigate a misfire too soon. Follow the regulations and local laws set for this purpose or, if none are available, wait a prudent time.

In case of misfire, do not drill or otherwise handle an explosive charge without instructions from a competent and experienced person and who expressely is authorized to solve this type of problem.

Organize all work so that the blasts coincide with the moments when least personnel is present, and try to always use the same time of day.

35.3.10 *Safety precautions with misfires*

Place warning signs on misfired blastholes.

The situation must be corrected before recommencing drilling in proximate areas.

If the round was electric and the circuit is visible, check to see if it is continuous from a safe distance and fire it if all is correct, take extra precautions for possible flyrock.

In case of priming with detonating cord, try to remove the stemming material and place a primer cartridge next to the explosive for its destruction. Stem the round with sand or a fine granular material.

If the explosive cannot be reached, drill a new hole at a distance of over 10 *D*, when the local laws and regulations in force permit.

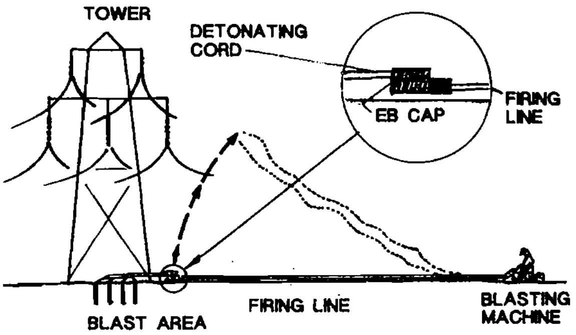


Fig. 35.24. Recommended initiation system when firing blasts in areas that are very close to power lines.

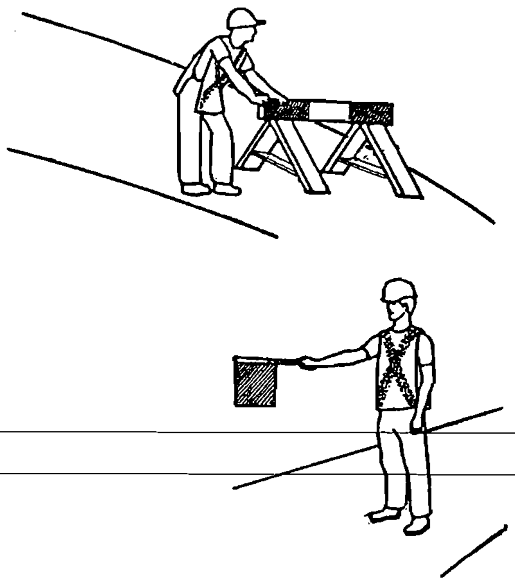


Fig. 35.25. Control all accesses to the blast area.

Always use highly specialized personnel for the task of neutralizing and eliminating explosives that have not detonated.

35.3.11 *Safety measures with secondary blasting*

Check to see if cracks or fissures are visible on the surface of the boulders.

Place the boulders in an area where there is a shielding effect against airblast; for example, at the bottom of a stable wall.

Always try to use the blasthole method because the system of patch explosives or plaster shooting produce higher levels of airblast and noise.

35.3.12 *Measures to be taken when discarding explosives*

You should be aware that deteriorated or damaged explosives may be more dangerous than those in good condition.

Always destroy or discard explosives in compliance with the approved methods: by combustion, explosion and by chemical destruction. Maintain the prescribed distance.

When destroying by explosion, it is recommended that this be done inside a blasthole, under sand or water, because if it is carried out in the open the airblast and noise would be extremely high. Initiation should be electric with proper primers.

When combustion is used, the amount of explosive per round should not go over 15 kg. To the contrary, it should be carried out in various locations.

Never burn explosives in their original packing bags or boxes. Once extended over a surface, they can be sprayed with gasoline to favor combustion.

When various combustion operations are carried out, each one should be done in a different location.

Never allow paper, carton of other materials used to pack explosives to be burned in fireplaces, heaters or other closed places, nor used for other purposes. Place them in thin layers in an appropriate area and in the open air, and when burning, a distance of at least 30 m must be kept.

The chemical destruction, which is one of the methods used for blasting agents, especially ANFO, is carried out by dissolving the nitrates in water. In these cases, precautions must be taken to control contamination.

When discarding detonating cord, the best procedure consists of laying it out in straight lines in a bed of dry fire wood or straw, wetting it with gasoline or fuel oil, as with conventional explosives. It should never be burned on the reel.

The electric detonators and milisecond delay caps are destroyed, if in large quantities, confined in a cavity or hole in the ground with the help of some explosive or by wrapping detonating cord several times around the bunch of accessories.

REFERENCES

Anonimo: *Normas y Sistemas de Seguridad en los Túneles.* Noticias Perfosa, no 2, Revista Minera, Febrero, 1987.

Atlas Powder CO.: *Electrical Hook-Ups Surface Blasting.*

B.O.E.: *Instrucciones Técnicas Complementarias.* 11 de Abril de 1986.

Encasur: *Medidas de Seguridad. Perforación y Voladura.* 1982.

Estudios y Proyectos Mineros: *Manual de Seguridad en Minería a Cielo Abierto.* ITGE, 1991.

EXSA: *Siempre-Nunca lo que Vd. Debe Saber al Usar Explosivos.* Perú, 1985.

Gadner-Denver: *Drilling with your Gardner-Denver Rotary Rig.*

Ingersoll-Rand: *Safety Manual.*

Langa, E.: *Precauciones en la Perforación y Carga de Explosivos.* Normas de Seguridad en el Uso y Manejo de Explosivos. INSHT, Octubre, 1984.

Pavetto, C.S.: *Surface mine blasting – A program guide for certification.* McGraw Hill, Inc., 1986.

Union Española de Explosivos, S.A.: *Transporte de Explosivos por Carretera.*

Union Española de Explosivos, S.A.: *Prescripciones que se han de Poner en Práctica en los Supuestos de Rotura o Deterioro de Envases y Embalajes.*

Union Española de Explosivos, S.A.: *Destrucción de Explosivos Industriales.*

Conversion factors

Length

~~mm (millimetres)~~	~~× 0.001~~	~~= m~~
cm (centimetres)	× 0.01	= m
dm (decimetres)	× 0.1	= m
km (kilometres)	× 1000	= m
in (inches)	× 25.4	= mm
ft (feet)	× 0.305	= m
yd (yard)	× 0.914	= m
miles	× 1609	= m

Power

kW (kilowatts)	× 1000	= W
Horsepower, metric	× 735.5	= W
Horsepower, U.K.	× 745.7	= W
ft.lbf/sec	× 1.36	= W
Btu/h	× 0.29	= W

Volume

l (litres)	× 0.001	= m^3
ml (millilitres)	× 0.001	= l
dm^3 (cubic decimetres)	× 1.0	= l
cm^3 (cubic centimetres)	× 1.0	= ml
mm^3 (cubic millimetres)	× 0.001	= ml
~~in^3 (cubic inches)~~	~~× 16.39~~	~~= ml~~
ft^3 (cubic feet)	× 28.316	= l
Imperial gallon	× 4.546	= l
U.S. gallon	× 3.785	= l
Ounces (Imp.fluid oz)	× 28.41	= ml
Ounces (U.S.fluid oz)	× 29.57	= ml
Pints (U.S. liq.)	× 0.4732	= l
Quarts (U.S. liq.)	× 0.9463	= l
yd^3 (cubic yards)	× 0.7646	= m^3

Force

kN (kilonewton)	× 1000	= N
kp (kilopond)	× 9.81	= N
kgf (kilogramme force)	× 9.81	= N
lbf (pound force)	× 4.45	= N

Torque

kpm (kilopondmetres)	× 9.81	= Nm
lbf in (poundforce/inch)	× 0.11	= Nm
lbf ft (poundforce/foot)	× 1.36	= Nm

Mass (weight)

g (grammes)	× 0.001	= kg
t (tonnes, metric)	× 1000	= kg
~~grains~~	~~× 0.0648~~	~~= g~~
oz (ounce)	× 28.35	= g
lb (pounds)	× 0.4536	= kg
tons (long, US)	× 1016	= kg
tons (U.K.)	× 1016	= kg
tons (short)	× 907	= kg

Speed

km/h (kilometres/hour)	× 0.2777	= m/s
m/s	× 3.6	= km/h
mph (miles/hour)	× 0.45	= m/s
mph	× 1.61	= km/h
ft/s (foot/second)	× 0.3048	= m/s
ft/s (foot/second)	× 18.29	= m/min
ft/min (foot/minute)	× 0.3048	= m/min

Frequency

blows/minute	× 0.017	= Hz
kHz (kiloHertz)	× 1000	= Hz
r/min.(rev./minute)	× 0.01667	= r/s
~~degrees/second~~	~~× 0.1667~~	~~= r/min~~
radians/second	× 0.1592	= r/s

Pressure

bar	× 100	= kPa
bar	× 100 000	= Pa
kp/cm^2	× 0.98	= bar
atm (atmospheres)	× 1.01	= bar
psi (pounds/in^2)	× 6.895	= kPa
~~psi~~	~~× 0.06895~~	~~= bar~~

Area

mm^2 (square mm)	× 0.000001	= m^2
cm^2 (square cm)	× 0.0001	= m^2
in^2 (square inches)	× 645	= mm^2
ft^2 (square feet)	× 0.929	= m^2
yd^2 (square yards)	× 0.8361	= m^2
Acres	× 4047	= m^2
Square miles	× 2.590	= km^2

General information, weight of materials

Weight$^{1)}$ of materials	bank lb/cuyd/kg/m^3	swell in %	swell factor	loose lb/cuyd/kg/m^3
Basalt	**5000/2970**	**52**	**0.66**	**3300/1960**
Bauxite, kaolin	**3200/1900**	**33**	**0.75**	**2400/1420**
Carnotite, uranium ore	**3700/2200**	**35**	**0.74**	**2750/1630**
Cinders	**1450/ 860**	**54**	**0.65**	**950/ 560**
Clay – natural bed	**3400/2020**	**22**	**0.82**	**2800/1660**
dry	**3100/1840**	**24**	**0.80**	**2500/1480**
wet	**3500/2080**	**24**	**0.80**	**2800/1660**
Clay with gravel – dry	**2800/1660**	**17**	**0.86**	**2400/1420**
wet	**3100/1840**	**19**	**0.84**	**2600/1540**
Coal – anthracite, raw	**2700/1600**	**35**	**0.74**	**2000/1190**
washed		**35**	**0.74**	**1850/1100**
ash, bituminous coal	**1000-1500/590-890**	**8**	**0.93**	**900-1100/550-830**
bituminous coal, raw	**2150/1280**	**35**	**0.74**	**1600/ 950**
washed		**35**	**0.74**	**1400/ 830**
Decomposed rock 75% rock, 25% earth	**4700/2790**	**42**	**0.70**	**3300/1960**
50% rock, 50% earth	**3850/2280**	**33**	**0.75**	**2900/1720**
25% rock, 75% earth	**3300/1960**	**24**	**0.80**	**2650/1570**
Earth – dry packed	**3200/1900**	**26**	**0.79**	**2550/1510**
wet excavated	**3400/2020**	**26**	**0.79**	**2700/1600**
loam	**2600/1540**	**23**	**0.81**	**2100/1250**
Granite – broken	**4600/2730**	**64**	**0.61**	**2800/1660**
Gravel – pitrun	**3650/2170**	**12**	**0.89**	**3260/1930**
dry	**2850/1690**	**12**	**0.89**	**2550/1510**
dry 6-50 mm (¼"-2")	**3200/1900**	**12**	**0.89**	**2850/1690**
wet 6-50 mm (¼"-2")	**3800/2260**	**12**	**0.89**	**3400/2020**
Gypsum – broken	**5350/3170**	**75**	**0.57**	**3050/1810**
crushed	**4700/2790**	**75**	**0.57**	**2700/1600**
Hematite, iron ore	**4700-6400/2130-2900**	**18**	**0.85**	**4000-5400/1810-2450**
Limestone – broken	**4400/2610**	**69**	**0.59**	**2600/1540**
crushed				**2600/1540**
Magnetite, iron ore	**5500/3260**	**17**	**0.86**	**4700/2790**
Pyrite, iron ore	**5100/3030**	**18**	**0.85**	**4350/2580**
Sand – dry, loose	**2700/1600**	**12**	**0.89**	**2400/1420**
damp	**3200/1900**	**12**	**0.89**	**2850/1690**
wet	**3500/2080**	**13**	**0.88**	**3100/1840**
Sand with clay – loose	**3400/2020**	**26**	**0.79**	**2700/1600**
compacted				**4050/2400**
Sand with gravel-dry	**3250/1930**	**12**	**0.89**	**2900/1720**

wet	3750/2230	10	0.91	3400/2020
Sandstone	4250/2520	67	0.60	2550/1510
Shale	2800/1660	33	0.75	2100/1250
Slag – broken	4950/2940	68	0.59	2950/1750
Snow – dry				220/ 130
wet				860/ 520
Stone-crushed	4500/2670	67	0.60	2700/1600
Top soil	2300/1370	44	0.69	1600/ 950
Trap rock – broken	4400/2610	49	0.67	2950/1750

1) **Varies with moisture content, grain size, degree of compaction, etc. Tests must be made to determine exact material characteristics.**

Glossary

Abutment height: Distance from tunnel floor to point where side wall joins roof.

Advance per round:Length excavated per round in tunnels, shafts, etc.

Airblast:Overpressure of air produced during a blast.

Amplitude: Maximum displacement of a particle during ground vibration.

Anchor bolt: Metal piece of anchorage for reinforcing rock.

ANFO: Blasting agent in powder form consisting of Ammonium Nitrate and fuel oil.

ANFOPS: Mixture of ANFO with expanded polystyrene beads.

Axial priming: System of initiation for an explosive charge by means of a detonating cord placed lengthwise along the charge.

Bench blasting: Type of blast that is carried out with one or two free faces on a bench using vertical or horizontal blastholes.

Bench: A basic unit of operation consisting in a rock shelf with a vertical plane or face, and a horizontal plane or work platform.

Black powder: Compound or mixture of saltpeter, sulphur and carbon which ignites abruptly at a certain temperature, giving off a large quantity of gases.

Blast, blasting: Fragmentation of rock or other solids by means of explosives confined in blastholes or placed against their surface.

Blaster, shotfirer: A person who is qualified for charging and supervising blasts.

Blasthole pressure: Pressure exerted by hot detonation gases against the blasthole walls.

Blasthole, borehole: Cylindrical opening drilled in rock for lodging explosives.

Blasting circuit: Electrical circuit used to initiate one or more detonators.

Blasting machine: Equipment used to energize electric blasting caps or other special ones.

Blasting agent: A mixture of combustible and oxidizing substances that are not of an intrinsically explosive nature by themselves.

Bottom charge: Explosive, normally of high strength, placed at the bottom of a blasthole.

Boulder: Rock fragment which is too large to be handled by the loading equipment.

Bubble energy: Energy of the gases produced by an explosion. It is usually measured in underwater tests.

Burden: Distance from blasthole to free face of the rock, measured perpendicularly to the axis of the hole.

Cartridge: Explosive cartridge in cylindrical form.

Cast blasting: Blasting for maximum rock displacement or throw.

Centralizer: Drilling accessory used to make the drill string rotate on the same axis in order to reduce deviation.

Column charge: An explosive placed between the stemming and the bottom charge.

Collaring: Point on the surface where the blasthole drilling begins.

Charge concentration: Quantity or weight of explosive per lineal meter of blasthole for a given diameter. Also expressed per unit of area in contour blasts.

Confinement: Degree of burial of an explosive charge inside the rock.

Connector: Blasting accessory used to unite the detonating cords or wires of the electric blasting caps.

Contour, perimeter. The final profile or surface planned in an excavation.

Controlled blasting: Blasting technique used to control overbreak and produce a competent slope. It can be presplitting, smooth blasting, cushin blasting, etc.

Coupling: Metal piece which joins the drill rods.

Crater : Cone shaped hole made in the rock by an explosive charge placed inside.

Critical diameter: Diameter below which a stable detonation of an explosive charge will not take place.

Crown pillar: Upper part of ore chamber being exploited and below the drilling level.

Cushion blasting: Shot fired after the main blast using decoupled charges with little spacing in order to produce competent slopes.

Cut: Section opened in the rock to create a free face in blasts of great confinement, such as in tunnels.

Cutoffs: Part of charged blasthole is cut off by detonation of adjacent hole with lower delay number.

Cuttings: Particles or chips of rock produced when drilling a blasthole.

Charge weight per delay: The sum of the explosive charges considered to detonate in an interval of under 8 seconds within a blast.

Dead pressure: Pressure which causes the achieved density of an explosive to loose its sensitivity for detonation.

Deck charge: A charge made up of bulk explosive, cartridges or fractions of either, separated by inert material or air. The initiation can be instantaneous or delayed.

Decoupling: Separation between the surface of an explosive charge and the blasthole wall where it is lodged.

Deflagration: Rapid, shockless subsonic explosive reaction accompanied by the formation of gases and blasthole pressure.

Delay element: Initiation element that is used to dephase in time the moment of detonation of a charge with respect to another.

Demolition: Breakage and knock down of an artificial structure by blasting.

Density: Relationship between the mass and the volume of a body such as rock, explosives, etc.

Detachment: Separation produced between fragmented and remaining rock after a blast.

Detonating cord: Plastic covered cord filled with a explosive of high strength and velocity, used to initiate explosive charges and transmit detonation.

Detonation velocity: Propagation velocity of the detonation wave through an explosive charge.

Detonation pressure: Pressure measured in the C-J plane, behind the detonation front, during propagation through an explosive column.

Detonation: Explosive reaction consisting in propagation of a shock wave through the explosive accompanied by a chemical reaction that frees a large quantity of hot, high pressure gases.

Detonator or electric blasting cap: Initiation accessory in capsule shape which, in the electric version, can be instantaneous, delay (0.5s) or millisecond delay (20 or 30 ms). They can also be classified according to intensity of impulse.

Dilution: Mixture of ore with waste or other worthless material.

Dip: Angle formed by blasthole inclination with the vertical.

Drag bit: Rotary, reamear shaped drill bit with cutting action and hard metal inserts along the edges.

Draw point: Excavation of inverted triangular section used for the extraction of fragmented ore in underground metal mines.

Drift (adit or dean): Lineal underground excavation, usually in mining.

Drill steel shank: Part of a drilling accessory that is between the hammer piston and the drill string.

Drill bit: Drilling tool which transmits the destructive energy to the rock.

Drill dust: Particles of rock produced when drilling blastholes.

Drilling pattern: Geometric plan for the placing of blastholes in a blast. It also includes their length, diameter and direction.

Drilling error: Deviation from the calculated blasthole pattern.

Drilling maneuver: Any operation that refers to any change or translation of the drilling equipment, lengthening of the drill steel, removing it, cleaning the blasthole, etc.

Drilling rate: Average blasthole drilling rate, including non-productive times.

Dynamite: Explosive invented by Alfred Nobel. At present, it is considered to be any explosive in which the sensitizer is nitroglycerine or a similar compound.

Echelon or staggered pattern: Placing the blastholes of one row in such a manner that they form triangles with those of the preceeding row.

Emulsion: Blasting agent in which the ammonium nitrate is dissolved in water and forms droplets surrounded by fuel oil.

Explosion: The mechanical process in which a mixture of gases, liquids and solids react instantaneously, forming hot, high pressure gases.

Explosive: Substance or compound that, by sudden release of its energy, produces an overpressure in the surroundings accompanied by flame and noise.

Extraneous electricity: Flows of electrical current that circulate outside a normal conductor. It can be the result of defective insulation of the equipment, the galvanic action of two different metals, etc.

Face: Free surface in a blast.

Feed: Component of rotary percussive rock drills upon which the pneumatic or hydraulic hammers move back and forth, also supplying the necessary thrust load for the advance.

Firing line: Electric cable used to connect the blast to the blasting machine.

Floor (foot wall): Lower surface of a bench, tunnel or chamber.

Fly rock: The hurling of rock fragments in a blast.

Fracture: Plane of discontinuity or breakage of rock inside the rock mass.

Fragmentation: Size of the pieces of blasted rock.

Frequency: Characteristic parameter of a vibration (H_z or cycles per second).

Fumes: Dangerous or harmful gases produced in a blast.

Half cast: Cylindrical surface or wall formed in the rock when drilling a blasthole.

Hammer drilling: Equipment with either pneumatic or hydraulic drive that is used to drill blastholes by transmission of energy to the rock in the form of shock waves.

Heavy ANFO: Blasting agent consisting in an emulsion mixed with ANFO.

Impedance: Characteristic which serves to measure the amount of energy transfered by a material. The product of the density is measured by sonic velocity.

Indentation: The act of detonating an explosive by means of a detonator or any other accessory.

Initiator: Blasting accessory that produces detonation.

Insert: Hard metal piece on drill bit.

Joints: Planes of weakness in the rock mass which offer no resistance to separation.

Jumbo: Self propelled equipment with one or several booms which have hammers for drilling in underground operations.

Lifters: Lower horizontal, or slightly inclined, blastholes which break upwards.

Loose rock: Rock fragments that come loose because of cracks, discontinuities, etc.

Mat: Materials used to control fly rock.

Microballoons: Small crystal or plastic spheres which are added to explosives to increase their sensitivity.

Millisecond delay: Dephase of time in milliseconds (less than 100 ms) with which diverse blasting accessories are manufactured, such as detonators, electric blasting caps, relays, etc.

Muck pile: After blast pile of fragmented rock.

Nitroglycerine (NG): $(C_3H_5 (ONO_2)_3)$.

Omega tube: Plastic tube opened by a generatrix that serves to prepare charges with decked cartridges in contour blasting.

Open pit: Surface operation by downward benching in beds of ore, inclined coal beds, etc.

Overbreak or backbreak: Rock breakage produced outside the thtical contour of a blast.

Particle velocity: Speed at which a ground particle moves from its rest position.

Particle acceleration: Parameter for measuring a unit of ground vibration in grams (1 g = 9.8 m/s^2).

Net penetration : Average rate of advance during blasthole drilling.

PETN: Pentrite. (Pentaerythritaltehanitrate)

Plaster shooting or popping: Secondary fragmentation or breakage of boulders produced during previous blasts.

Pneumatic loader: Equipment run by compressed air. It is used for charging either bulk or cartridged explosives.

Pocket priming: Initiation system carried out by placing the primer cartridge on the bottom or top of the explosive column.

Powder magazine: A portable or permanent structure used to store explosives and blasting accessories.

Presplitting: Contour blast consisting in blastholes which break before the stoping blasts and create a cut or fissure plane in the rock.

Prill: Small, porous sphere of ammonium nitrate.

Primer cartridge: An explosive charge of high strength and sensitivity into which the detonator is placed. It is used to potentiate and increase the output of other explosives.

Quarry: Surface operation for the extraction of rock for ornamental and construction purposes: lime, marble, granite, and shale are examples.

Radiofrequency energy: Electric energy transmitted through the air in the form of radio or electromagnetic waves.

Raise: Underground upward excavation of a tunnel or shaft that can be vertical or with an inclination of approximately 45° from the horizontal.

Retrac bit: A drill bit which has grooves and teeth on the back part which permit backward drilling in poor ground which tends to cave in.

Rip-rap: Large sized blocks of rock used for construction of dams, piers, channel protection, etc.

Ripping blast: Blast with very little displacement, which helps increase natural fracturation or rock swell when preparing for posterior breakage and loading.

Round: Blast, or amount of rock excavated in one blast.

Row of holes: Series of aligned blastholes in a blast.

Scaled distance: Relationship used to predict the vibration level and air blast. It is usually expressed by distance in meters, from the blast to the point of observation,divided by the square or cube root of the charge weight per delay, in kilograms.

Scaling: Removal of rock that is loose or in poor condition from the side walls and faces.

Secondary blasting: Blasting technique used to control overbreak and produce a competent slope. It can be presplitting, smooth blasting, cushion blasting, etc.

Seismograph: Instrument that measures and permanently registers ground vibrations induced by earthquakes and blasting.

Sensitivity: Measure of susceptibility of an explosive to detonate when it receives an external subsonic impulse. Also a measure of its ability to propagate the detonation.

Sensitizer: Ingredient used in explosives to increase facility of initiation or propagation of the reactions.

Sequential blasting machine: Blasting machine which has diverse channels through which current is discharged with a prefixed dephasing of milliseconds.

Shaft: Vertical underground excavation that is either of rectangular or circular sections and of larger dimensions than raises.

Shank adaptor: First piece on the drill string which transmits the energy from piston impact and breakage to the drill string gears.

Shaped charge: An explosive prepared with a special geometrical configuration used fundamentally in secondary fragmentation and demolitions.

Shock tube system (Nonel): An initiation system for detonators in which energy is transmitted to them by means of a shock wave that travels through a plastic tube.

Shock wave: A wave that propagates through the explosive charge at supersonic velocity in that medium.

Shot: Charged blasthole.

Side wall: The lateral face of an excavation.

Slope: Relationship between the horizontal projection and the height of the bench face.

Smooth blasting: Contour blast made up of blastholes placed close to each other with weak charges, fired after stoping blasts to achieve smoother profiles.

Spacing: Distance between blastholes of the same row.

Spalling: Mechanism of rock breakage that is produced when the shock wave is reflected on a free face as a consequence of the great difference in impedances.

Specific drilling: Meters of blasthole or volume drilled per ton or m^3 of rock broken.

Specific charge: Amount of explosive used per ton or m^3 of blasted rock.

Square pattern: Placing the blastholes on the vertexes of a square.

Stemming: Areas of blasthole without explosive charge that are filled with inert material such as dust, sand, clay, etc.

Stope: Steplike excavation underground for the removal of ore that is formed as the ore is mined in successive layers

Strength: Property of an explosive measured by diverse methods, which express its working capacity.

Stress, or strain energy: Energy transmitted by the shock wave generated by an explosive.

Strip mine: Surface operation with only one bench in which the bottom usually coincides with the wall of the ore level.

Subdrill: Length of blasthole drilled below the planned level of breakage.

Sublevel: Drifts opened at different levels to exploit ore beds.

Swelling: Increase in rock volume after breakage.

Sympathetic detonation: Detonation of an explosive material by means of an impulse from the detonation of another charge, sent through air, ground or water.

Tamping stick: Wooden or plastic tool used to insert and tamp explosive cartridges and stemming material in a blasthole.

TNT: Trinitrotoluene

Toe: Unfragmented parts of a rock mass which appear in the lower areas of the bench.

Tricone (rolling cone) bit: Rotary drill bit with crushing and ripping action, made of three cones with steel teeth or hard metal inserts.

Tunnel: Lineal underground excavation.

Underwater blasting: Rock blasting underwater.

Vibration: Oscillating movement of a solid or liquid medium provoked by passage of an elastic wave.

Watergel: Slurry composed of an aqueous solution of ammonium nitrate, which has been sensitized with fuel oil and to which a crosslinking agent has been added, turning the consistency from fluid to elastic.

Subject index

Abrasion value, 6, 31
Abrasiveness, 6
~~Absolute weight strength, 100~~
Abutment height, 218
Advance per round, 219
Air
    deck presplitting, 264, 265, 266
    deposit, 89
Airblast, 278, 339, 340
Aluminum, 109, 111, 114, 115
Ammonium nitrate, 106
Amphibolite, 6
Amplitude, 339
Anchor bolt, 252, 261
Angled cuts, 227
ALANFO, 109
ANFO, 106, 107, 108, 109
ANFOPS, 259, 260
Angular deviation, 30, 190
Attenuation, 339
Axial priming, 137, 138

Ballistic mortar, 98
Basalt, 3, 6
Beams, 315
Bearing, 65
Bench, 182, 191-203
    blasting, 182, 191-203
    floor, 295
Bit penetration, 30, 31, 32, 33, 34
~~Black powder, 116~~
Blast
    blasting, 154, 155, 156, 157, 158, 159
    geometry, 179
Blastability index, 167, 168
Blaster, shot firer, 379, 380
Blasthole
    borehole, 179-190
    deviation, 190
    inclination, 181, 192, 194
    pattern, 183, 199
    pressure, 187, 257
Blasting
    agent, 106-115
    caps, 123
    circuit, 127, 128, 129
    machine, 131, 132
    shelter, 370
Booster charge, 136
Boreability, 4, 31, 32, 33
Borehole, 179-190
    pressure, 187, 257
Boring, 82
Bottom
    charge, 187
    priming, 140
Boulder, 173, 295, 299, 300, 301, 302
Breakage by flexion, 156
Breaking agent, 301
Brick walls, 313
~~Bridges, 317~~
Bridgewires, 128
Brisance, 58
    values for explosives, 58
Brittleness Test, 31
Bubble energy, 157
Buffer blasting, 255
Building
    damage, 350
    demolition, 312
    response, 350
Bulk
    explosives, 106, 136, 145-151
    loading trucks, 147
Burden, 183, 199, 200, 201, 202, 203
Burn cuts, 225, 226
Button bits, 41

Carbon dioxide, 95, 104
Cartridge, 142, 144
Cast blasting, 196
Centralizer, 190
Centralizing elements, 132
Chamber, 249
Channel effect, 104
Chapman-Jouguet (CJ), 93
Charge
    concentration, 188
    ~~configuration, 186, 192, 194~~
    diameter, 119
    trucks, 145
    weight per delay, 333
Chimneys, 316
Column charge, 187
Collar, 182
    flyrock, 366
Collaring deviation, 190
Combustion, 92
Compressive strength (confined and unconfined), 4, 160
Compressor, 87
Concrete, 313, 360
Conductivity, 161
Cone, 64
Confinement, 183, 337
Connector, 132
Contour
    blasting, 252-270
    holes, 252-261
    perimeter, 252-261
Controlled blasting, 252-270
Costs, 34, 61, 323
Coupling, 40, 186, 258
Covering, 368, 369
Crack length, 329, 330
Crater, 239
    blasting, 239, 240
    charges test, 99
    ~~cuts, 226~~
    volume, 239, 240
~~Cratering, 239, 240~~
Critical diameter, 103
Crown pillar, 242, 243
Crushing of rock, 154
Cushion blasting, 255
Cut, 219
    and fill mining, 248
Cutting
    action, 72
    tools, 75
Cuttings, 10, 52
Cutoffs, 285
Cylinder compression test, 99
Cylindrical cut, 219

Damage prevention criteria, 352-357
D'Autriche method, 101
Dead pressure, 104
Deck charge, 186
Decoupled charges, 186, 257, 258
Decoupling, 186
Deflagration, 92
    -to-detonation transition, 92
Delay
    element, 125
    timings, 285, 286, 287
~~Delayed blast, 281~~
Demolition, 312-322
Density, 102
    of rock, 160
Depreciation, 34, 62
Desensitization, 104
Detachment, 295
Detonation, 92
    cord, 127
    pressure, 102, 187, 257
    transmission, 103
    velocity, 101
~~Detonator or electric blasting cap, 127~~
Dewatering systems, 152
Diabase, 3, 6
Diesel oil, 107
Digital processing of images, 291, 292, 293
Dilution, 259
Dip, 163
Displacement, 290, 293
Double pipe test, 99
Down the hole hammer, 16, 17, 18
Drag bit, 72, 73
Drift (adit or dean), 217
Drifting and tunneling, 217-230
Drill
    bit, 41-47
    dust, 10, 52

  steel shank, 38
  string, 36, 53
Drillability index 31, 175
Drillhole, 179, 190
Drilling, 1, 8, 36, 48
  accessory, 36
  deviation, 30, 190
  error, 190
  maneuver, 33
  pattern, 183
  rate, 31
  rate index, 31
  test, 31, 59
Drop ball, 302
Dust collector, 28, 29, 55
Dynamite, 115

Echelon or staggered pattern, 183
Elastic modulus, 5, 330, 338
Elasticity, 5
Electric
  delay detonator, 127
  detonator, 127
Electronic detonator, 131
Emission of fumes from commercial explosives, 104, 122
Emulsion, 111, 112, 113
  explosive, 111, 112, 113
Enthalpy, 94
Environmental
  effects, 276, 333
  limitations, 306
Expansion volume, 186
Explosion, 92
Explosive, 92, 98, 106
  consumption, 188
Explosives for smooth blasting and presplitting, 259
Extraneous electricity, 128, 146, 378

Face, 163, 165
Fan
  cut, 227
  pattern, 245
Feed, 9, 56
Firing line, 128
Fixation factor, 201
Floor (foot wall), 218
Flushing, 10, 52, 75
Flyrock, 366
Foundations, 213, 313
Four-section cut, 219
Fracture, 163
Fracturing by release-of-load, 155
Fragmentation, 326
Free
  face, 184
  surface, 184
Frequency, 339
Fumes, 104
Fume class, 104
Funnel, 133
Fuse, 126

Gas extension fractures, 155
Gelatin dynamite, 115
Geophysical techniques, 170
Glass micro balloons, 111
Gneiss, 4, 6
  granite, 4, 6
Granite, 3, 6
Greywacke, 4, 6

Half cast factor, 267
Hammer drilling, 10-18
Handheld drills, 28
Hardness, 4
Heat
  of detonation, 94
  of formation, 94
Heats of formation of some explosives, 94
Heavy ANFO, 113
Hematite, 6
Hercudet non-electric initiation system, 124
High-speed camera photograph, 291
Hole deviation, 30, 190
Hydraulic rock drills, 5, 12, 14

Ice, 304
Igneous rocks, 3
Impedance, 156
Inclination, 181, 192, 194
Indentation, 8, 64
Initiator, 136
Insert bits, 41, 68
Inspection, 349
Instrumentation, 340, 341, 342
Internal friction, 161
Iron ore, 6
IADC codes, 71

Jet piercing, 83
Joints, 167
Jumbo, 20, 21, 22

Kaolin, 4, 6
Kodewimetro, 101
KUZ-RAM model, 327, 328, 329

Langefors weight strengths, 201
Lifters, 218, 222
Lifter holes, 218, 222
Limestone, 4, 6
Line drilling, 255
Linear
  charge concentration, 102
  charge concentration for conventional contour blasting, 262
Lithology, 161
Livingston theory, 239, 240
Livingston's Breakage Process Equation, 239, 240
Loading
  density, 102, 146
  equipment productivity, 293
Longhole blasthole mining method – LBH, 243
Loose rock, 290

Magnadet initiation system, 129
Magnetite, 6
Maintenance and repairs, 35, 62
Marble, 4, 6
Mast, 51
Mat, 369
Maximum
  cooperating charge, 334
  throw, 366
Mechanical notching tool, 267
Metamorfic rocks, 4
Microballoons, 111
Millisecond delay, 125, 126
Mine safety regulations, 371-381
Miniblast, 299
Mini-hole blasting, 214
Misfires, 380
Monocrystalline diamond, 75
Muck pile, 293
Multiple-row round, 185, 282

Net penetration, 30, 59
Nitrocellulose, 115
Nitrogen oxides, 104
Nitroglycerine explosives, 95, 115
Nitroglycerine (NG), 95
Noise, 339, 357
Non-nitroglycerin explosives, 106
NONEL, 123
Nonideal detonation, 93
Noxious gases, 104
Nozzle, 68

OD method, 77
ODEX method, 77
Omega tube, 132, 258
Open pit, 172, 191
Optimum depth ratio, 239
Ornamental rock, 268
Overbreak or backbreak, 268, 294, 295
Oxygen
  balanced explosives, 95
  deficiency, 95

P-wave, 338
  velocity, 338
Parallel hole cut, 219
Particle
  acceleration, 339
  velocity, 339
Peak
  particle velocities, 339
  vibration particle velocity, 339
Pegmatite, 6
Penetration rate, 30, 59
Percussion, 8
Permissible explosives, 116
PETN, 127
Photogrammetric method, 290
Photographic method, 290
Pillars, 315
Piston compressor, 87
Planning, 305
Plaster shooting or popping, 299
Plasticity, 6
Plate dent method, 99
Pneumatic
  charger, 145
  drilling rigs, 10, 11
  loader, 144, 145, 146
Pocket priming, 140
Polycrystalline diamond, 75
Pop shooting, 299
Porosity, 160
Powder
  factor, 188
  magazine, 375
Practical burden, 201
Preblasting, 215
Predicting the fragmentation, 326
Presplitting, 254
Pressure
  drop, 90
  of the explosion, 96
Prill, 107
Primer, 136
  cartridge, 137
Probabilistic analysis, 331
Propagation Law, 342-345
Pulldown, 56, 57

  hoisting systems, 50
Pump trucks for explosives, 149
Pumpable blasting agent, 114, 199
Pumping emulsion explosive, 111, 113
Pyrotechnic delay detonator, 125

Qualitative visual analysis, 290
Quarry, 191

Radial fracturing, 154
Radio frequency energy, 128, 146, 378
Raise, 231
  driving, 80
Ramp blasting, 210
Rayleigh wave, 338
Reaction products of some explosives, 94
Relative weight strength, 100
~~Retrac bit, 41~~
Rip-rap, 195, 266
Ripping blast, 216
Rock
  breakage mechanisms, 154
  constant, 201
  properties, 160
Rolling cone rock bits, 64, 71
Room and pillar mining, 248
Rotary
  power, 58
  rig, 48-62
  screw, 87
Rotation systems, 49, 50
Round, 105
Row of holes, 185
RQD, 167
RQI, 174

SABREX system, 329
Safe distance, 257, 369
Safety
  fuse, 126
  measures, 371-381
Salt mines, 228, 229
Sandstone, 4, 6
~~Scaled distance, 342~~
Scaling, 373
Schistosity, 4, 163
Screening, 293
Secondary blasting, 293, 299
Sedimentary rocks, 4
Seismic
  strength test, 98
  survey, 170
Seismograph, 340, 341
Sensitivity, 102
Sensitizer, 104, 110
Sequential blasting machine, 131
Shaft, 231
  sinking, 79
Shank adaptor, 37
Shaped charge, 278, 299
Shinking cut, 210
Shock
  subs, 54
  tube system (Nonel), 123
  wave, 154, 159
Shot, 105
Side
  ~~hill cut, 206~~
  wall, 218, 252
Slate, 4, 6
Sliding-vane compressor, 88
Slope, 252, 358
Slurry explosives, 110, 111
Smooth blasting, 254
Spacing, 183
Spalling, 154
Specific
  charge, 188
  drilling, 189
  heat, 94
Square pattern, 183
Stability, 102
Staggered pattern, 183
Steel, 36
Stemming, 182
  height, 182
Stope, 240
Stoping holes, 239-248
Strain
  energy factor, 239
  wave, 156
Stray currents, 128, 146, 378
Strength, 98, 154
Stress
  or strain energy, 157
  fields, 164
  waves, 154, 253, 358
Strip mine, 191
Stump, 302
Subdrilling, 182
Sublevel stoping, 245
Surface coal mines, 172, 196
Swelling, 290
Sympathetic detonation, 103

Tamping stick, 133
Temperature of the explosion, 96
Tensile strength, 160
Texture, 6
Thermochemistry, 94
Threads, 36
TNT, 95, 110
Toe, 295
Top priming, 141
Towers, 317
Toxic fumes, 104
~~Trauzl Lead Block Test, 98~~
Trench blasting, 208
Tricone (rolling cone) bit, 64-71
Tungsten carbide inserts, 68
Tunnel, 217

Unconfined compressive strength, 4, 160
Underground stopes, 239, 284
Underwater
  blasting, 272-279
  explosion tests, 100
  shock waves, 276

V-cut, 227
Vacuum filter, 89
VCR blasting, 236, 242
Velocity of Detonation (VOD), 101
Vertical Crater Retreat Blasting technique (see VCR), 236, 242
Vibration, 333-365
  level, 339
Volume of explosion, 95
VOD measurement, 101

Water resistance, 102
Watergel, 110, 111
Water-jet drilling, 84
Wave parameters, 338, 339
Wedges, 301
Weight strength, 100
  relative to LFB-dynamite and ANFO for some explosives, 100

Young's modulus, 330, 338